JOURNAL OF

METALS

December 1954

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JOURNAL OF

METALS

VOL. 6, NO. 12 DECEMBER, 1954

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COVER

New techniques and equipment have resulted in a steady progress toward economic vacuum melting. For this special issue Harry Brocke has illustrated the dominant characteristics of vacuum metallurgy as suggested from photographs furnished by the F. J. Stokes Machine Co. and by the Consolidated Vacuum Co.

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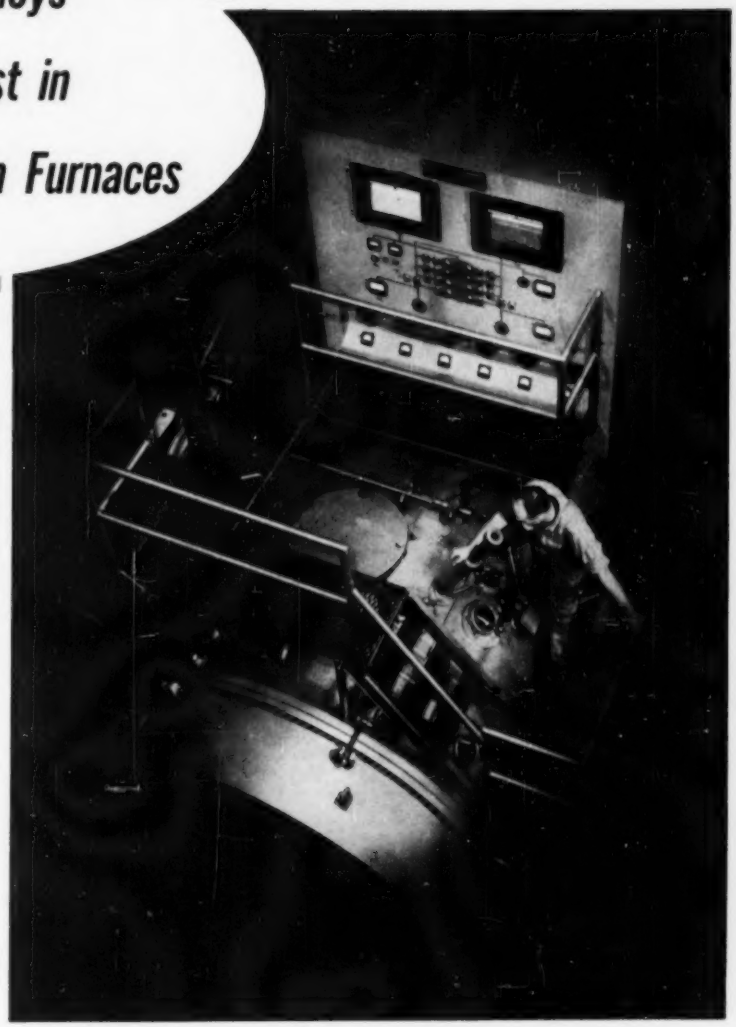

A Stokes high-vacuum melting furnace of 1000-pound capacity at Utica Drop Forge & Tool Corporation, Utica, N. Y. The furnace is to be used for the melting and casting of high-temperature alloys for jet engine rotor blades.

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Meet The Authors

Gerhard Derge (p. 1386) is on the staff of the metals research laboratory, Carnegie Institute of Technology, Pittsburgh. Dr. Derge was born at Lincoln, Neb., and graduated from Amherst College and Princeton University. Since 1934 he has been a staff member at Carnegie. Several articles have been published by AIME. During his free time, Dr. Derge enjoys fishing and farming.

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D. J. CARNEY

S. GILBERT

S. Gilbert (p. 1383) joined the applied research laboratory of the U. S. Steel Corp., Pittsburgh in 1951. Born at Pittsburgh, Mr. Gilbert graduated from the University of Pittsburgh, B.S., metallurgical engineering. From 1948 to 1951 he was on the staff of the metals research laboratory, Carnegie Institute of Technology. His hobby is his home work shop.

E. C. Rudolphy (p. 1391) graduated from Carleton College, Northfield, Minn., in June 1949. He began with U. S. Steel Corp., South Works, in 1949 as a metallurgical trainee. From 1950 to 1952 he was a metallurgical research technologist and since March 1952, Mr. Rudolphy has been petrographer. He is a Member of AIME, ASM, and American Ceramic Society. He has written articles for the *Blast Furnace, Coke, and Raw Materials Proceedings* and also JOURNAL OF METALS.

E. C. RUDOLPHY

W. O. PHILBROOK

W. O. Philbrook (p. 1396) was born in Chicago and graduated from the University of Chicago. Following graduation he joined the Wisconsin Steel Works of the International Harvester Co., Chicago. He joined this company as a test carrier and became foreman of the metallurgical div. Since 1954 he has been associated with Carnegie Institute of Technology. Mr. Philbrook was assistant professor of metallurgical engineering from 1945 to 1948, and was named associate professor in 1948. He is also senior research metallurgical engineer in the metals research laboratory. Mr. Philbrook has authored several transactions articles and was co-editor on the second edition of *Basic Open Hearth Steelmaking*. In 1944 he received the McKune Award and was a co-winner of the R. W. Hunt Award in 1952. He has been active in the Iron and Steel Div.

M. T. Simnad (p. 1386) is a member of the research laboratory, Carnegie Institute of Technology. Dr. Simnad was born at Teheran, Iran, and is a graduate of London University and Cambridge University. He has previously had articles published in the JOURNAL OF METALS.

Frank Arthur Forward (p. 1408) professor and head of the dept. of mining and metallurgy was born at Ottawa, Canada. He attended the Ottawa Collegiate Institute and University of Toronto. He has been on the staff of the University of British Columbia since 1935. Prof. Forward has been granted several patents and others are pending. He holds membership in various societies both in the U S. and Canada, and has had many papers published.

J. Halpern (p. 1408) was born in Poland. He received his degrees from McGill University, Montreal, Canada. In 1950 he joined the staff of the University of British Columbia as assistant professor of metallurgy. He is co-author of articles previously published by AIME.

I. George (p. 1386) has been head of the technical dept., British Industrial Solvents, div. of Distillers Co. Ltd., Glamorgan, Wales, since September 1952. Mr. George was born in Wales and attended Swansea University College, (B.Sc.). In July 1943 he joined the G.K.B. Iron & Steel Co. Ltd., Cardiff, South Wales as metallurgical assistant. He remained with this firm until March 1945, when he accepted a position with the British Industrial Solvents, calcium carbide factory, as a technical assistant. In August 1950 Mr. George traveled in the United States on an ECA fellowship. He returned to Wales in September 1952. Mr. George is a member of the British Iron & Steel Institute.

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THE following employment items are made available to AIME members on a non-profit basis by the Engineering Societies Personnel Service Inc., operating in cooperation with the Four Founder Societies. Local offices of the Personnel Service are at 8 W. 40th St., **New York 18;** 100 Farnsworth Ave., **Detroit;** 57 Post St., **San Francisco;** 84 E. Randolph St., **Chicago 1.** Applicants should address all mail to the proper key numbers in care of the New York office and include 6c in stamps for forwarding and returning application. The applicant agrees, if placed in a position by means of the Service, to pay the placement fee listed by the Service. AIME members may secure a weekly bulletin of positions available for $3.50 a quarter, $12 a year.

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Mining or Metallurgical Engineers, graduates, 25 to 45, experienced as mill shift bosses or experienced in metallurgical research work. Favorable climate; modern housing available. Apply by letter stating age, educational background, experience, salary requirements and when available. Location, West Coast. W618-S.

Mill Metallurgist Assistant, young, with several years experience in ore separation. Single status. Salary open. Location, Guiana. F593.

Engineers. (a) Welding engineer, process, methodize and tool up production of stainless steel, inconel, aluminum aircraft parts. Close tolerances. Heliarc, arc, acetylene, nicrobraze, resistance welding. Specify and procure jigs and fixtures. Engineering degree preferred; five years specific experience required. (b) Stampings engineer, process, methodize and tool up production of stainless steel, inconel, aluminum aircraft parts. Close tolerance, deep draws. Kirksite dies, Whistler and Wale setups. Presses 2T to 450T. Specify and procure dies, fixtures. Engineering degree preferred; five years specific experience required. Salaries open. Location, Midwest. W574.

Engineers. (a) Production process engineer, degree or equivalent, 25 to 55, for manufacturer of jet engines. Must have had at least four years experience in basic sheet metal equipment, hydraulic, O.B.I. gap and horan presses—power brakes, rolling, expanding, and seizing equipment. Must have knowledge of presses and dies, functional type dies for piercing, blanking, forming and drawing operation. Duties will be processing of sheet metal parts (stainless steel) for jet aircraft engines. Salary, $6000 or better. (b) Production process engineer, degree or equivalent, 25 to 55. Must have had at least four years experience in production process experience in both fusion (inert gas shield arc) and resistance (spot and seam welding) experience in the use of three-phase resistance welding equipment, fusion and resistance welding. Duties will be determining the need for functional tooling and equipment necessary for this type of processing. Salary, $6000 or better. Location, Chicago. C2181.

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Industrial Notes

• *Aluminum Co. of America's research laboratory at New Kensington, Pa., has a team working on a study of porcelain enamel on aluminum. This porcelain enamel-type finish may be characterized as a vitreous oxide coating that is fused to an aluminum base.*

• **Mackintosh-Hemphill, Pittsburgh, is building a 48-in. automatic roll contouring lathe for the new 24-in. wide flange beam mill now under construction at the Indiana Harbor plant of Inland Steel Co. The electronically-controlled duplicating lathe will be able to turn and redress steel mill rolls up to 48-in. diam and 18 ft long. Operating from a stylus following a flat master template, cutting tool on the lathe will machine structural-shape contours on the roll. Raytheon, Inc., Waltham, Mass., will supply electronic contouring equipment. General Electric Co. will build the main drive motor, auxiliary motors, and controls.**

• More than $15 million has been saved in the cost of aircraft engine cylinder walls since the Navy adopted a process for chromium plating the walls of engine cylinders. The process has extended engine cylinder service life from 1200 to more than 3000 hr of operation.

• Kaiser Aluminum & Chemical Corp.'s $27 million expansion program at Baton Rouge, La., is nearing completion. When finished, the plant's capacity will be more than 800,000 tons.

• *The Atomic Energy Commission has adopted a policy of encouraging unclassified research on uranium. Studies of constitution diagrams and crystal structure, electronics structure, mechanisms of corrosion, thermodynamics, phase transformations, electrochemical studies, and other basic research on uranium and its alloys can be undertaken as unclassified research. Work may be carried out under these general principles: any subject for research must have the approval of the AEC's Office of Classification and a license must be obtained for purchase of natural uranium if more than 3 lb is required. A project leader must generally be given a security clearance; and technical reports, although nominally unclassified, must be reviewed by the AEC before publication. Uranium itself is unclassified, and need only be accounted for on the basis of its monetary value. The AEC has also indicated willingness to provide funds to worthy projects or supply necessary small amounts of uranium in return for information developed.*

• *A series of conferences designed to acquaint engineers with the properties and applications of ductile iron has been started by International Nickel Co., Inc. Individual meetings, attended by design engineers, materials engineers, purchasing agents, and management representatives, are being held for companies using cast or wrought components. First of these conferences were held at the Worthington Corp., Harrison, N. J., and at M. W. Kellogg Co., New York City.*

• **Opening of a new laboratory specializing in fatigue testing of materials, and repeated loading of parts and components was announced by Krouse Western Laboratories, Inc., Van Nuys, Calif. Laboratory equipment includes facilities for direct stressing in tension or compression, flexual stressing, rotating beam loading, combined loading, and special setups for component testing.**

• *Electro Metallurgical Co., a Union Carbide & Carbon Corp. div., has renewed a $5000 grant to Iowa State College to support a research fellowship in the field of basic metallurgy. The project is under the direction of F. H. Spedding. Stanley P. Thompson has been named to receive the fellowship for the coming year.*

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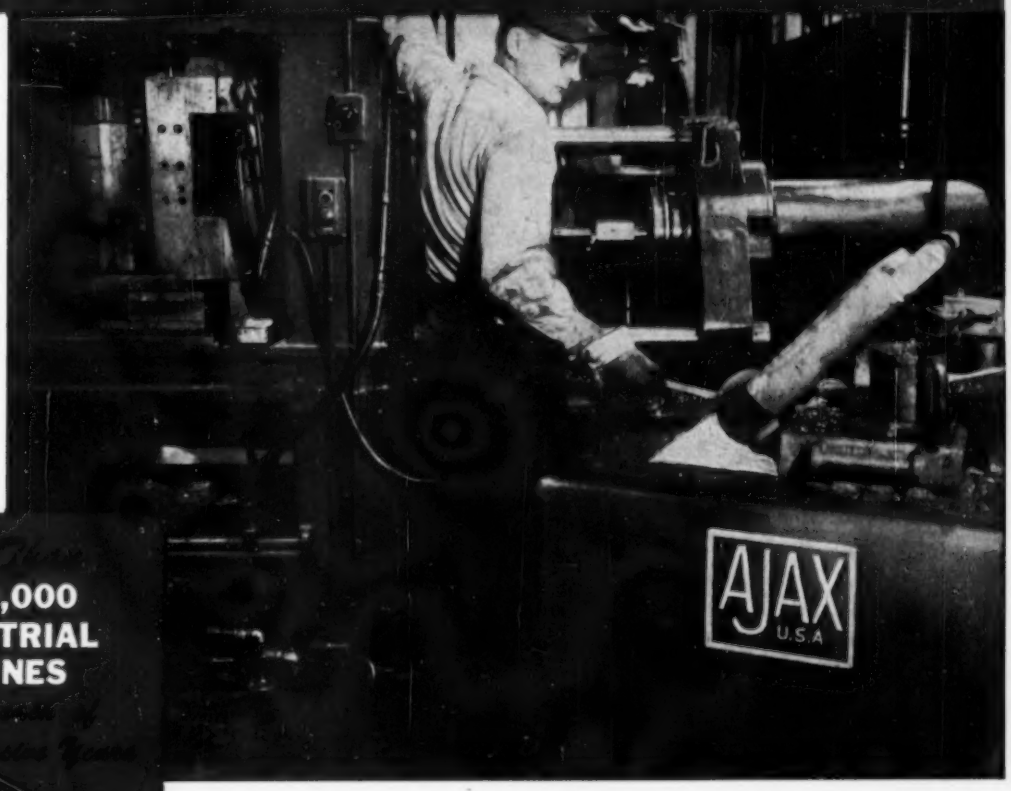

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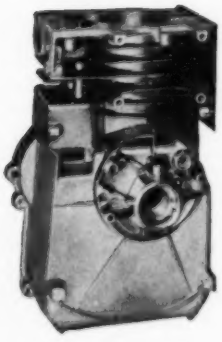

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This is another in a series of advertisements depicting the ways in which Ramtite Refractories can be of service in the steel industry.

— Coming Events —

Dec. 1-3, AIME, Electric Furnace Steel Conference, William Penn Hotel, Pittsburgh, Pa.

Dec. 3-4, AIME, Columbia Local Section, joint session with North West Mining Assn.

Dec. 6, AIME, Boston Local Section, Faculty Club, M.I.T., Cambridge, Mass.

Dec. 12-15, American Institute of Chemical Engineers, annual meeting, Statler Hotel, New York, N. Y.

Dec. 26-31, American Assn. for the Advancement of Science, national meeting, University of California, Berkeley, Calif.

Dec. 28-29, American Assn. for Advancement of Science, scientific manpower conference, Berkeley, Calif.

Jan. 12, 1955, AIME, Connecticut Local Section, Bridgeport, Conn.

Jan. 13, AIME, Cleveland Local Section, annual dinner meeting, Manger Hotel, Cleveland, Ohio.

Jan. 18, Society for Applied Spectroscopy, Philadelphia Section, Philadelphia College of Pharmacy and Science, Philadelphia.

Jan. 23-25, Indian Institute of Metals, annual general meeting, United Club, Jamshedpur, India.

Feb. 8-9, Midwest Welding Conference, Armour Research Foundation, Illinois Institute of Technology, Chicago.

Feb. 10, AIME, Cleveland Local Section, American Room, Manger Hotel, Cleveland.

Feb. 14-17, AIME, annual meeting, Hotel Conrad Hilton, Chicago, Ill.

Mar. 9, AIME, Connecticut Local Section, Bridgeport, Conn.

Mar. 10, AIME, Cleveland Local Section, American Room, Manger Hotel, Cleveland.

Mar. 10-11, Porcelain Enamel Institute, Pacific Coast conference, Biltmore Hotel, Los Angeles, Calif.

Mar. 14-15, Steel Founders' Society of America, annual meeting, Drake Hotel, Chicago, Ill.

Mar. 15, Society for Applied Spectroscopy, Philadelphia Section, Philadelphia, Pa.

Mar. 16, AIME, Connecticut Local Section, annual meeting, Statler Hotel, Hartford. Conn.

Mar. 20-23, American Institute of Chemical Engineers, Kentucky Hotel, Louisville, Ky.

Mar. 28-Apr. 1, Western Metal Exposition, Western Metal Congress; Pan-Pacific Auditorium, Ambassador Hotel, Los Angeles, Calif.

Apr. 14, AIME, Cleveland Local Section, joint meeting with American Ceramic Society, NACA Laboratories, Cleveland Hopkins Airport.

Apr. 18-20, AIME, Blast Furnace, Coke Oven and Raw Materials and National Open Hearth Steel Conferences, Bellevue-Stratford Hotel, Philadelphia.

Apr. 19-21, Canadian Institute of Mining and Metallurgy, annual meeting, Royal York Hotel, Toronto.

Apr. 28-30, AIME, Pacific Northwest Conference, Spokane, Wash.

May 1-4, American Institute of Chemical Engineers, Shamrock Hotel, Houston, Texas.

June 1-18, Joint Metallurgical Societies, European meeting.

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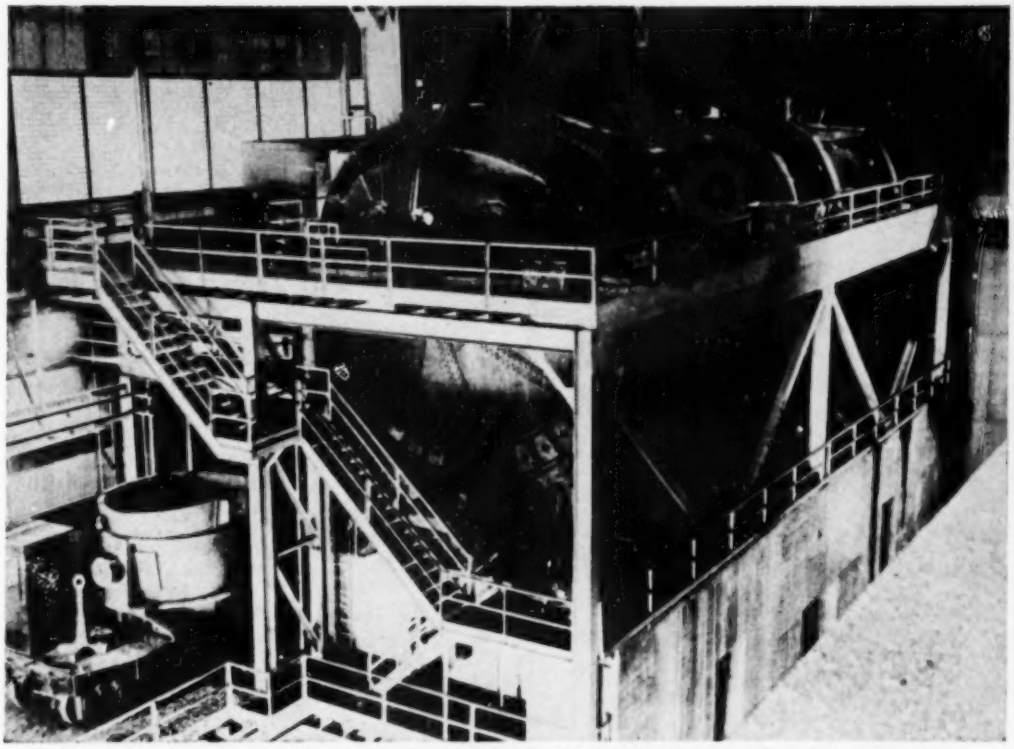

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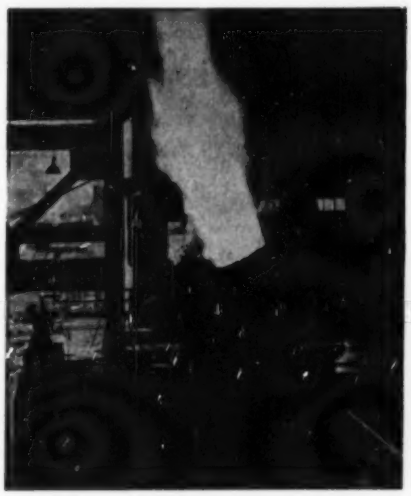
Tennessee Coal & Iron Division, U. S. S.
Bethlehem Steel Company
Algoma Steel Company, Canada
Youngstown Sheet and Tube Company
Republic Steel Corporation
Siderurgica del Mediterraneo, Spain
Inland Steel Company
Monterrey Steel Company, Mexico
Cia Acos Especiais Itabira, Brazil
Great Lakes Steel Corporation
National Steel, Brazil
Corporacion de Fomento, Chile
Cia Siderurgica Belgo Mineira



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Ladle Transfer Cars
Cupolas
Slag Cars
Dragouts
Slag Pots
Jib Cranes
Jack Cars
Bottom Oven Cars
Bottom Ovens
Post Ladle Cranes
Ingot Cars
Scrap Buckets
Charging Box Cars
Blast Furnaces

This advertisement is appearing in the 1955 edition of the
WATKINS CYCLOPEDIA OF THE STEEL INDUSTRY

Books for Engineers

ORDER YOUR BOOKS THROUGH AIME—Address Irene K. Sharp, Book Department. Ten per cent discount given whenever possible. Order Government publications direct from the agency concerned.

Welding for Engineers, by Harry Udin, Edward R. Funk, and John Wulff. *John Wiley & Sons, Inc.* $7.50, 430 pp., 1954—This book offers the reader a thorough appreciation of welding as an art and a science. Welding is treated as a field itself, rather than an offshoot of some other discipline. Drawing on principles from physics, mathematics, chemistry, metallurgy, and mechanical engineering, the authors carefully construct an understanding that extends not only to welding and joining but to the other arts in the general field of metal fabricating. The emphasis is on principles rather than practice. However, its presentation also equips the engineer with information and an approach that will enable him to deal with a variety of actual welding problems.

Chemical Engineering and Equipment Construction, Monographs No. 275 to 282, Vol. 23, by Herbert Bretschneider and Kurt Fishbeck. *Deutsche Gesellschaft fur chemisches Apparatewesen.* Approx. $6.40, 242 pp., 1954—This volume contains the text of a series of lectures by Kirschbaum students, delivered at the suggestion of the Dechema on the 25th anniversary of the Institute of Chemical Apparatus, Equipment and Processing.

Six Papers on the Development of Chemical Apparatus and Equipment, Monographs Nos. 269 to 274. Vol. 22, by Herbert Bretschneider and Kurt Fishbeck. *Deutsche Gesellschaft fur chemisches Apparatewesen.* Approx. $4.25, 139 pp., 1954—The volume covers the first lecture meeting of the European Federation for Chemical Engineering in 1953. Full text of the lectures are in German with English and French summaries.

Shell Molding and Shell Mold Castings, by T. C. Du Mond. *Reinhold Publishing Corp.* $2.00, 128 pp., 1954—Despite the fact that the author admits to certain difficulties in compiling this volume, enough sound information is available and presented here to permit the engineer to learn how it works and the advantages of the process. It is not the purpose of the book to supply enough knowledge to permit someone to go out and set up in business. Its purpose is to permit evaluation of the method.

Nickel in Iron and Steel, by A. M. Hall. *John Wiley & Sons, Inc.* $10.00, 596 pp., 1954—This is the second in the new series of Alloys of Iron Monographs. The introduction is concerned with occurrence and refining of nickel ores. The remainder of the book contains material on such topics as: the physical properties of steels containing nickel; the structure and heat treatment of wrought and cast nickel steels; and the effect of nickel on various engineering properties. Corrosion and welding of nickel steels are discussed in detail. Other important subjects treated in this volume are the effect of nickel on the constitution of cast iron, its tensile strength, and its hardness. Each chapter is concluded with a summary.

Fundamentals of Electromagnetic Waves, by Paul C. Shedd. *Prentice-Hall Inc.* $5.65, 191 pp., 1954—The book is intended to aid in the familiarization of students of electrical engineering and physics with the fundamental principles of electromagnetic fields and to lay a basis for study of electromagnetic waves. Effort has been made to place emphasis upon fundamentals, rather than to present a superficial treatment of the subject. No mathematical knowledge has been assumed beyond that of elementary calculus.

Thermodynamics, by Franklin P. Durham, *Prentice-Hall, Inc.* $7.35, 312 pp., 1954—This book was written because of the author's dissatisfaction with other texts used on the college level for the teaching of thermodynamics. The intent of the book is to: develop basic theory in terms of a minimum number of simple equations; present simply the requirements for a necessity of reversible processes in the analysis of ideal thermodynamic processes; introduce the concept of entropy early in the text and to integrate this concept throughout, along with the treatment of irreversible processes; introduce the concept of stagnation conditions in steady flow processes and to stress the importance of their use; and use a set of symbols incorporating a system for the distinction between entire and specific quantities, and quantity rates—with no double meanings of their use.

Verhütten Von Eisenerzen, second edition, (German), by Robert Durrer. *Verlag Stahleisen M. B. H.*, 159 pp., 1954—The book deals in a comprehensive manner with the disadvantages of the blast furnace smelting process and ways of preventing them. It also suggests new methods for producing iron.

The Physics of Particle Size Analysis. *The Institute of Physics.* Approx. $4.90, 218 pp., 1954—Papers in this volume were drawn from a conference arranged by the Institute of Physics, London, and held in the University of Nottingham from Apr. 6 to 9, 1954. Topics covered at the conference and in the book are: relative motion of particles and fluids: size separation; relative motion of particles and fluids: molecular phenomena; scattering and absorption of light by particles; particle shape factors; visual counting and sizing of microscopic particles; automatized counting and sizing; theory; and automatized counting and sizing: photoelectronic machines. The final session was a general one. The summary of the conference and concluding remarks are offered by H. Heywood, of the Imperial College of Science and Technology, London.

The Cupola and Its Operation, *American Foundrymen's Society.* $9.50, 332 pp., second edition, 1954.—A combined work by an impressive list of authorities that should be of value to gray iron foundries, which are operating more than 5000 cupolas in the U. S. and Canada. New material covers such late developments as hot blast, basic lining for nodular iron and emission control. Other chapters on refractories, principles of combustion and metallurgy have been greatly augmented. Bibliography, 328 illustrations, and 54 tables.

Please Order the Books Listed Below from the Publishers

The Ideal Foundry in Pictures, *Organisation for European Economic Co-operation.* $1.25, 64 pp., 33 illus., French and English, 1954.—A set of photographs showing U. S. foundry plants visited by a European mission. The whole range of foundry operations is presented with brief comments by a master foundryman. O.E.E.C. Mission, Suite 61, 2002 P St., N. W., Washington 6, D. C.

Non-Ferrous Heavy Metal Fabrication in the U. S. A., *Organisation for European Economic Co-operation.* $2.50, 260 pp., 53 illus.—A detailed study made in 1951 of U. S. techniques of fabricating sheet, strip, rods, sections, tubes, and wire of heavy nonferrous metals and alloys. The mission found that machines and methods employed here, while fundamentally the same as those used in Europe, are provided with a greater range of auxiliary equipment. This leads to higher machine utilization and in turn to greater productivity. O.E.E.C. Mission, Suite 61, 2002 P St., N. W., Washington 6, D. C.

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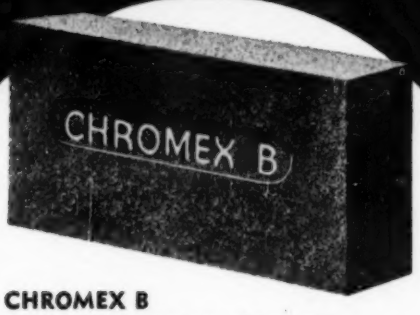

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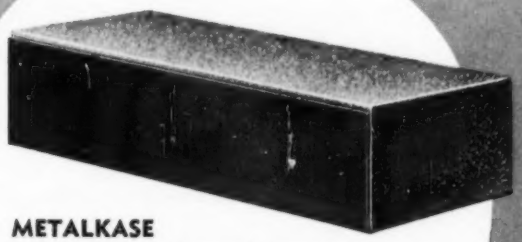

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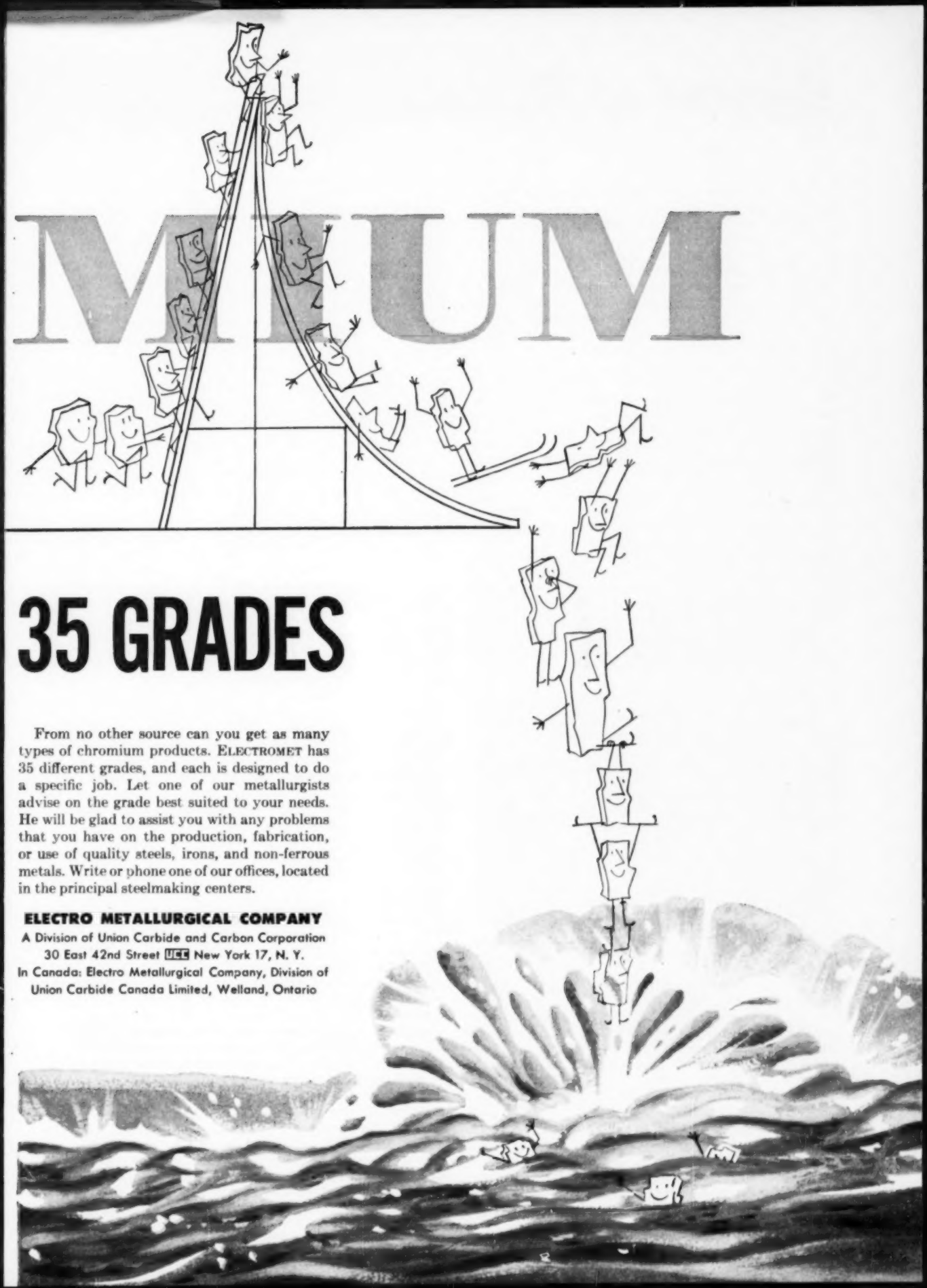

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ONE of Sweden's leading producers of stainless sheet and tube, Nyby Bruk, has installed a plant using the Rossi process for the continuous casting of steel billets and slabs. Until now ingots were reduced to billets and slabs in a conventional blooming mill. The new machine, built in Nyby Bruk's own shops, is the result of many years of research in the U. S., Canada, England, and at Nyby Bruk.

Molten steel is transported from the furnace in charges to the 60 ft high plant and tapped into a vertical water-cooled metal mold. Billets are gripped at the bottom end and automatically removed from the mold at the same pace as casting proceeds on top. Billets are also cut into desired lengths. Three men are required for the operation of the plant. Similar plants are in operation in other steel producing countries. However, the Swedish unit is reported to embody several improvements.

Under agreement with the U. S. firm owning the basic patents, Continuous Metalcast Corp., Nyby Bruk has rights for the entire world with the exception of the U. S. and Canada. During the period it was building its own plant, the Swedish company delivered units to firms in England and Yugoslavia. Another facet of the agreement between Continuous Metalcast and Nyby Bruk is the future exchange of all developments in the field of continuous casting. The Swedish firm is participating in work on a horizontal machine which has already been successfully employed by an American aluminum company.

Details of the Rossi process were carried in the JOURNAL OF METALS, March 1951.

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SOMETHING new is making it possible to use low alloy or mild steels instead of costly high alloy steels. Jet aircraft components, ships' boilers, hot air tubes and economizer elements, diesel exhaust pipes and mufflers, and petroleum refinery equipment parts are examples of possible application of the new Alumicoat process now being employed by Arthur Tickle Engineering Works. The process is being used under license from General Motors Corp. patent 2569097.

The process promises protection for steel and its alloys from both chemical and elevated temperature corrosion. Pure aluminum and its alloys are metallurgically bonded to the ferrous materials.

First step in the process is the cleaning of the surface to be bonded. Areas not to be coated are masked off. The parts are then fluxed in a salt bath and immersed in molten aluminum for a few minutes, producing a metallurgical iron-aluminum alloy bond at the intersurface. Aluminum penetrates the steel to a depth of −0.0015 in. to −0.001 in. and about −0.0005 in. to −0.001 in. aluminum clings to the exterior of the piece as it is extracted from the bath. Time of immersion and material used determines the coating thickness. Pure aluminum or any of its alloys can be used for the coating material. However, pure aluminum has been found to be the best for resistance to industrial gases and corrosive media, especially those containing sulphur compounds. In one case, a quantity of boiler economizer elements that had formerly required frequent replacement, were still in service 18 months after aluminum dipping.

High temperature oxidation of mild or low alloy steels can be stopped or materially reduced by the Alumicoat process. Tests show that adequate protection against oxidation is provided at temperatures at least as high as the base metal has useful strength. Low carbon steel tests were made at temperatures in excess of 1900°F. At temperatures exceeding the melting point of aluminum, the aluminum diffuses into the steel, forming an intermetallic compound of iron and aluminum. Well known as a refractory material, this diffused coating gives the steel maximum protection against high temperature sealing.

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LAST January 10 a Comet I jet airliner exploded and crashed into the sea off Elba. The 29 passengers and the crew of six were killed. Another jet airliner crashed near Naples on April 8. An inquiry into the crashes has concluded that metal fatigue caused the first crash. The structure of the plane's pressurized cabin failed somewhere, perhaps near a window. The sudden exhaust of air threw some of the passengers into space before the entire plane exploded. It is believed that the second crash was caused by the same factors.

Britain was proud of the Comet, built by the de Havilland Co. Many think that progress in jet airliner construction both in the U. S. and England depends to a great extent on the inquiry. Four men engaged in what several experts felt was the most important jet aircraft investigation ever conducted. Presiding was Lord Cohen, a Lord of Appeal. Assisting him were Sir William Scott Farren, technical director for A. V. Roe & Co., aircraft manufacturers; William Jolly Duncan, professor aeronautics and fluid mechanics at Glasgow University; and Air Commodore, Allen H. Wheeler, commander of the Aeroplane and Armament Experimental Establishment of the Royal Air Force.

The ghost of the plane that went down near Elba was resurrected from the Ligurian Sea. Only officers and men of the Royal Navy thought the job could be done. Thousands of pieces of the plane were reclaimed from the ocean floor with the aid of television cameras. About 70 pct of the recoverable weight of the jet was brought to the surface and then fitted together on a wooden framework. Some of the pieces were no bigger than matchboxes. Twisted seats, bits of carpet, and battered hatches were placed in position. The engines, 80 pct restored, were also emplaced.

After months of testing that included investigation of other Comets in pressure tanks and in flight the experimenters still had no idea of what happened to the planes. Results of the study were compiled in a volume 4 in. thick and containing 120,000 words and 285 illustrations.

The Royal Aircraft Establishment ruled out the idea of design defects and the engines were declared not to be at fault. In addition, there is not the slightest doubt that the crews of either plane were everything other than they should have been.

The reconstruction of the jet proved that the Elba disaster started with a tremendous upward force generated inside of the cabin. Most of the passengers were thrown forward and upward. It is estimated that within one third of a second the cabin was empty. Some of the passengers were drawn out of the plane without touching it. Everything supports the theory of metal fatigue.

Sir Arnold Hall, who directed the investigation, states that nothing in the investigation contradicts the metal fatigue theory. He noted that while possible to make accurate calculations of the requisite strength of standardized materials, a structure still could fail under stress repeatedly applied. There is no certain way for detection of metal fatigue.

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DIE casting's versatility and scope is exhibited by the wide range of shapes and sizes possible by this method. The American Zinc Institute believes improved product design and reduced costs are possible by use of the die casting method and is out to prove it. A movie, Die Casting—How Else Would You Make It?, is the Institute's latest vehicle for the dissemination of die casting information. The film presents a comprehensive evaluation of zinc, aluminum, magnesium, and copper base die casting alloys, outlining the advantages and adaptability of each metal.

A detailed examination of 45 outstanding examples of parts die cast from various alloys highlights the design flexibility of die casting in producing complex shapes within close dimension limits. The film records the latest design developments and production methods. One sequence illustrates how an automobile defroster duct composed of several individual zinc die castings is formed, assembled, and shipped as a complete unit by the die caster. The film may be booked from the American Zinc Institute, Inc., 60 East 42nd St., New York 17, N. Y.

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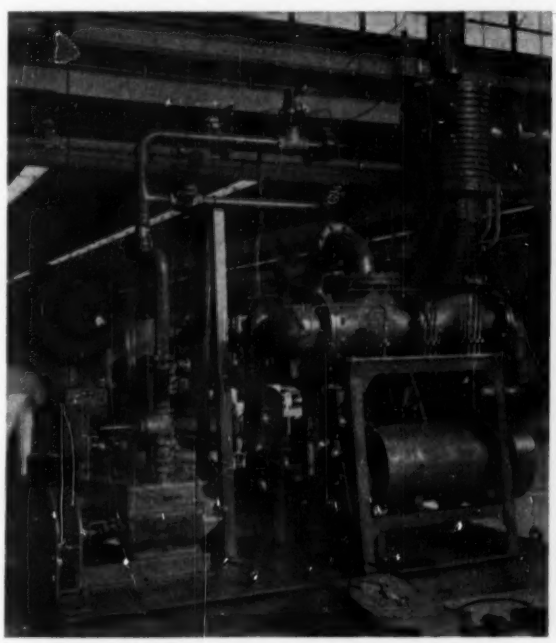
SOME time ago Freeport Sulphur Co. announced the discovery of a tremendously significant nickel deposit in the Moa Bay district of Cuba. With the agreement recently signed between the General Services Administration and Freeport the first big step toward developing the orebody into a significant contributor to the free world nickel supply is about to be made. The new agreement provides for the construction and operation by the company of a pilot plant to study on a large scale the new process for production of nickel and cobalt from Moa Bay.

Financed by the Government, the plant will be located near New Orleans on land purchased by Freeport. The unit will treat 50 tons of ore per day supplied by Freeport without charge to the Government. Work will be done by Freeport at no charge. The pilot plant is expected to be the forerunner of a commercial size unit. Freeport has already been financing study of certain phases of the process which include techniques devised by Chemical Construction Co.

Along with the pilot plant agreement, another contract provides for Government purchase of nickel and cobalt from the commercial plant which Nicaro Nickel Co., Freeport subsidiary, is planning to build at the successful completion of the pilot plant program. The purchase contract is contingent upon the Government's decision to enter into the commitment at the end of the pilot plant program. Possible purchase would be 150 million lb of nickel and 15 million lb of cobalt at prevailing market prices by June 30, 1963.

Nicaro Nickel plans to use private funds for the commercial plant. Part of the plant and auxiliary facilities may be constructed near the Moa Bay deposits. The ore would be leached with sulphuric acid and intermediate concentrate produced. The concentrates would be sent to the U. S. for separate extraction of nickel and cobalt metal. Production capacity contemplated is 30 million lb of nickel and 3 million lb of cobalt metal annually. The Moa Bay deposit is reported to contain 40 million tons of ore averaging 1.35 pct nickel and about 0.14 pct cobalt.



This two-stage diffusion-ejector pump, Consolidated Vacuum Corp. type KS-4000, is a recent pump development specifically for vacuum metallurgical processing. It's used in melting and casting furnaces having a crucible capacity of 350 to 500 lb of melt and can also be employed in the arc melting of 8-in. diam ingots.

Drift of Things

IT has recently been impressed on us how few people are aware of the many professional and educational interests and activities in which the AIME is directly or indirectly engaged. The Institute has never gone in for large scale publicity campaigns. It was considered sufficient that we are supporting these various projects—not for public acclaim—but for the genuine desire to advance the profession. Actually, we have been hiding our light behind a bushel, and perhaps it would be well for us to speak up more often. At this time we have singled out just one of the important activities that you should know more about.

AIME is one of five engineering societies that constitute the United States Engineering Trustees. UET coordinates the work and efforts of the societies, devoting itself to serve all engineering fields, and represents some 144,000 individual members of the societies involved. One of the divisions of UET, the Engineering Foundation, administered some 20 projects in the fiscal year ending Sept. 30, 1954. Four of these were AIME projects, and the following is a summary taken from the annual report.

Alloys of Iron Research continued its work on the new monograph series. Vol. 1, *Aluminum in Iron & Steel* was published early last year. Vol. 2, a comprehensive summary of data on nickel as an alloying element in steel and cast iron, was published in November. The manuscript for the third volume, *Titanium in Iron & Steel*, was edited for the press. In addition, five chapters dealing with columbium as an alloying element in steel were edited and sent out for review as part of the fourth volume, *Columbium, Tantalum, and Zirconium in Iron & Steel*. The rough draft has been completed for the fifth volume of the series, *Manganese in Iron & Steel*.

Difficulties in solving the complex problems in designing the equipment necessary for securing experimental data on *Heat Flow in Quenching* at Columbia University were overcome early in 1954, and a considerable number of quenching tests have been run. In quenching in still water the cooling rate changes rapidly with small changes in water temperature; if the water temperature rises above a certain value the vapor film on the specimen repeatedly breaks down and is re-formed. These, and other results of this research will undoubtedly prove of value in commercial quenching.

The important fundamental work on *Diffusion in Steel*, which is now in its seventh year at MIT, made substantial progress during the past 12 months. With the discovery, late in 1953, that the rate of diffusion during creep is not the same as in unstressed specimens, a strong emphasis has been placed on this phase of the diffusion problem. Further work along this line should have an important bearing on our theories of the high-temperature strength of metals and alloys.

Comminution Research at MIT completed its third year. During the present year, a method of fine-size analysis was developed by the use of centrifugal elutriation for fractionation. Size distribution was studied with the electron microscope. Work on the kinetic energy of a moving body has been continued; indications have already appeared that a short contact time between impacted and impacting masses should be beneficial in comminution, since a comparatively large amount of kinetic energy can be transferred to strain energy in the impacted material.

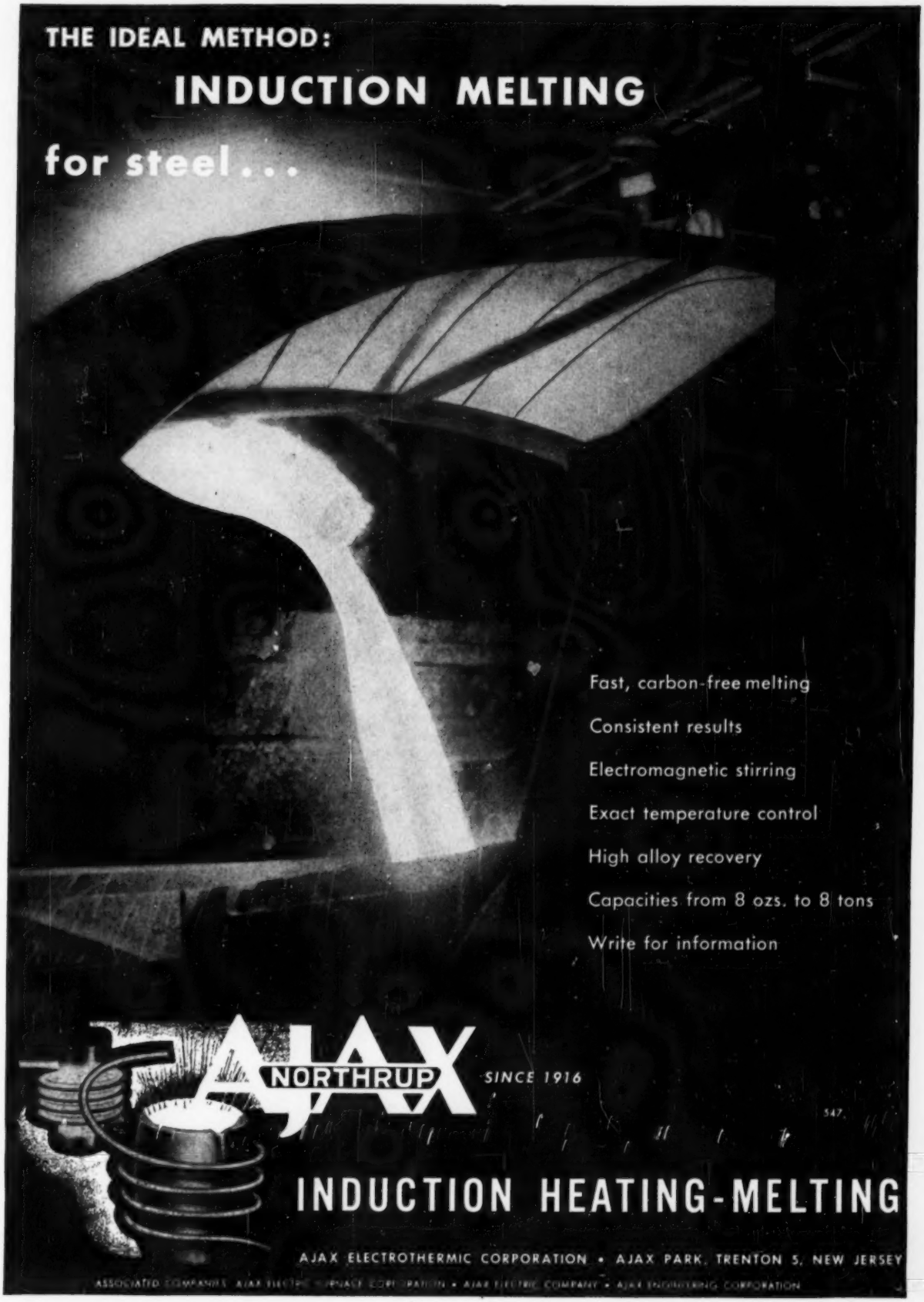
In addition, the Engineering Foundation continued its support of several joint society projects, the Engineers Council for Professional Development and the Welding Research Council, and supplied sponsorship and financial support of a project on the Evaluation of Engineering Education. The work of Engineers Council for Professional Development was extensive; the standing committees were active and made commendable progress. In the last year, 24,000 copies of the booklet, *Engineering—A Creative Profession*, were distributed, and a new addition of 75,000 copies is being printed. The other ECPD booklet, *After High School—What?*, has in the past year had a distribution of 40,000 copies. The Education Committee of ECPD has completed its reinspection of U.S. engineering curricula and has been requested to survey professional salaries, laboratory equipment and space, and other details in engineering colleges, to enable deans to compare their facilities with others. The special project for engineering training at the graduate level, started two years ago at Cincinnati with the aid of a special grant from the Foundation, has received excellent local support, and during the last school year, nearly 300 young men have been enrolled in courses at the University of Cincinnati.

Welding Research Council, now in its nineteenth year, supervised through its eight working committees about thirty research and other projects. The amount and quality of the work done by this internationally known group is evidenced continually by the technical papers prepared by its committee members, and its project directors and assistants. Welding Research Council has also extended its field of usefulness to industry during the year by setting up two new advisory committees—on aircraft and on public utilities—to survey, in cooperation with industry, the state of the art of welding in these fields, a survey which could logically lead to a cooperative effort to solve the problems uncovered by the survey.

The two year study of engineering education made by a committee of ASEE and 122 institutional committees appointed by the deans of all engineering colleges having accredited curricula, ended with a noteworthy report—presented to the ASEE at its annual meeting. The Committee made one recommendation which if implemented should have far-reaching consequences: that engineering curricula should be divided broadly into engineering-general and engineering-scientific. These two branches should have a common stem consisting of 100 semester hr of mathematics, basic science, engineering science, and humanities, with a little essential technology. The other 38 hr should consist of design, analysis, further science and technology, and options or electives.

This is but part of the work of a single division of UET.

A. S. Cohan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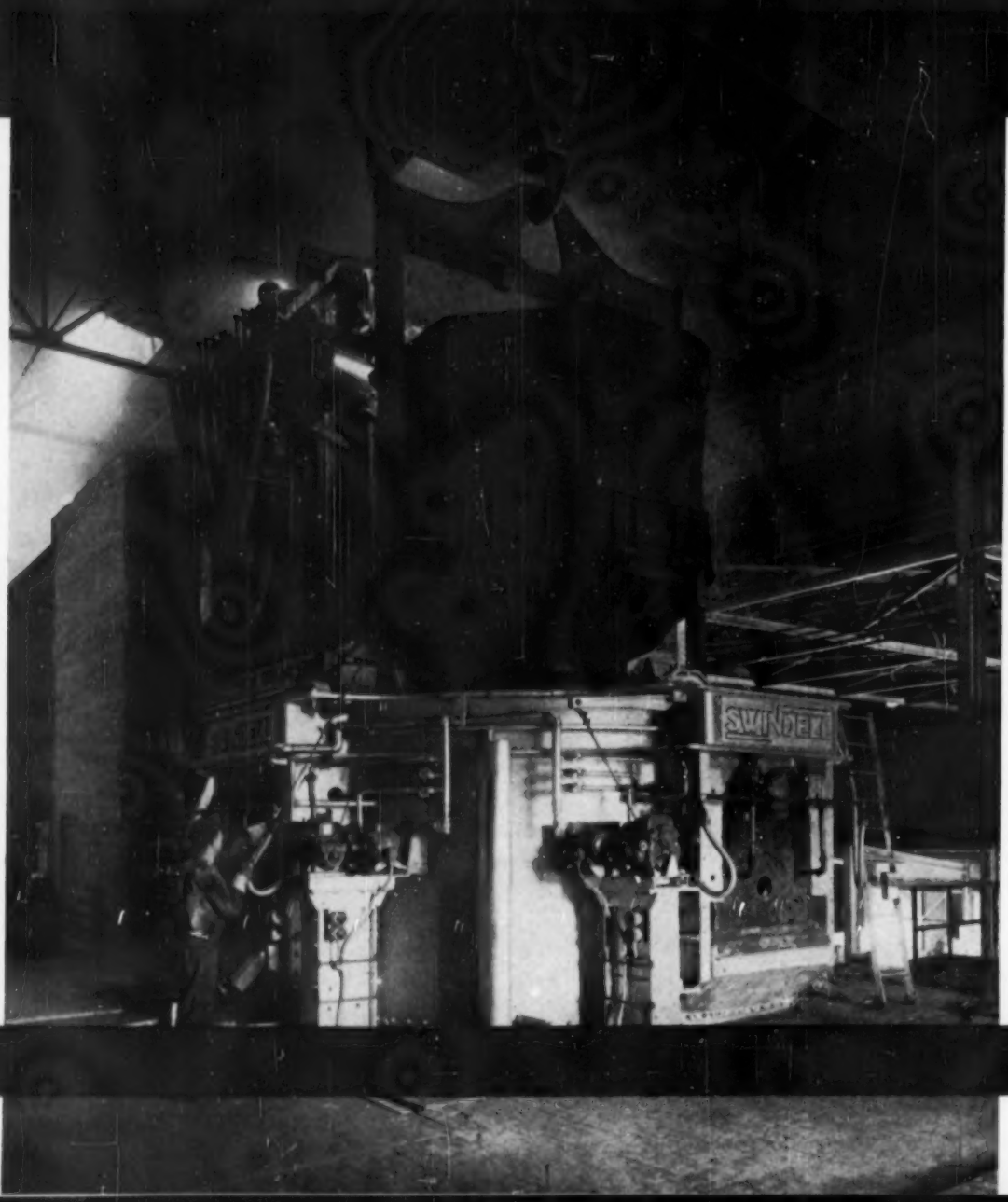

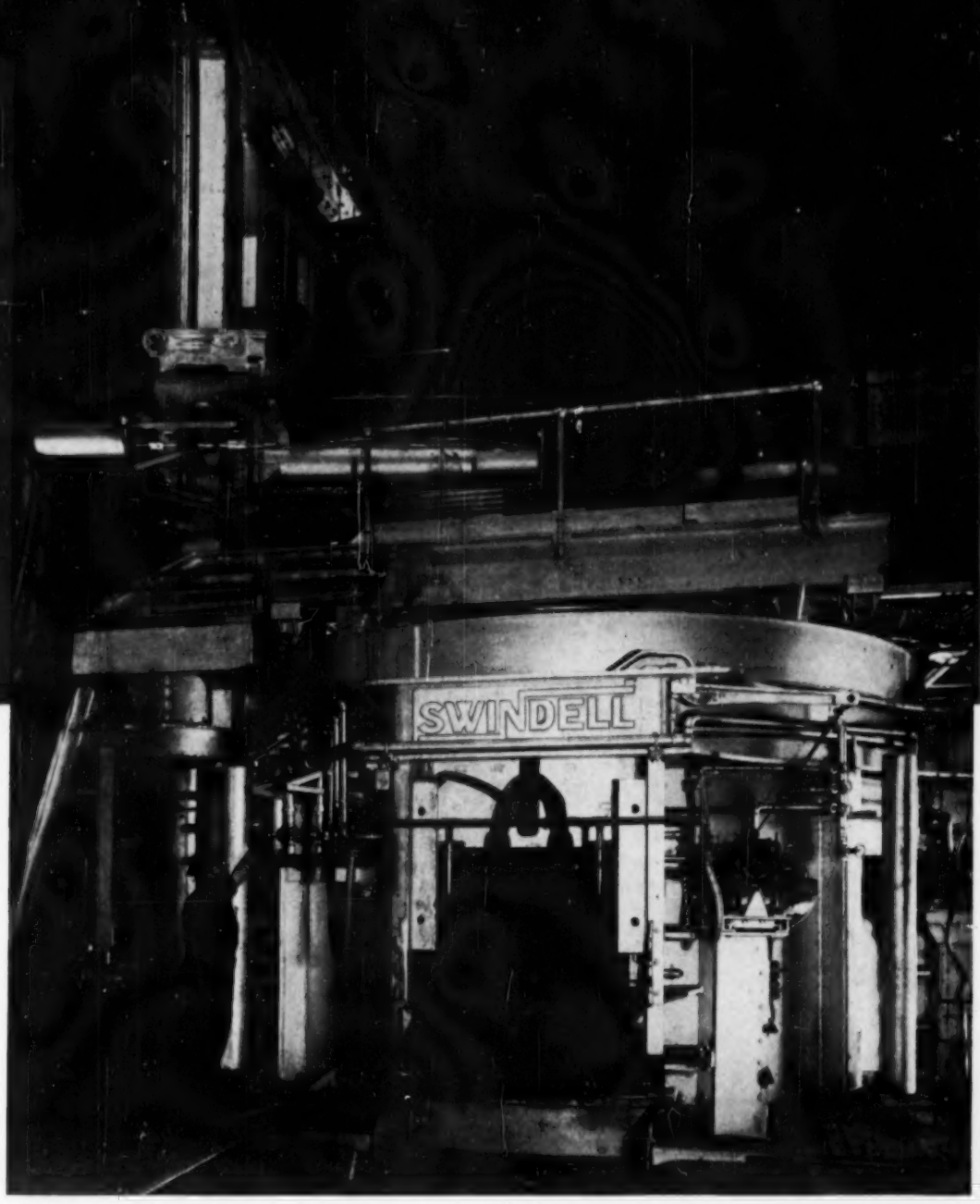

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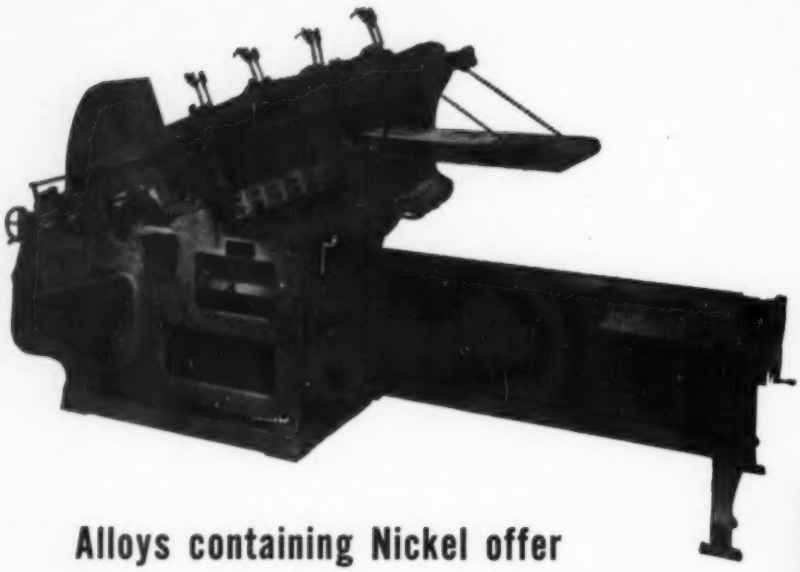

STAINLESS STEEL CONTAINING NICKEL resists attack by chemicals and atmospheres over a wide range of concentrations, pressures and temperatures. At elevated temperatures it resists oxidation, yet retains toughness and high strength down to —400°F. Easy to clean and keep clean, chromium-nickel stainless steel is used extensively by Package Machinery Co., of East Longmeadow, Mass., in the manufacture of the bacon packaging machine shown here and other food handling equipment.

Alloys containing Nickel offer

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SHOWN HERE are only four of hundreds of different combinations of properties. They give wider play to the skill of the engineer.

You can save weight and provide better durability, and frequently also improve response to fabrication . . . by using appropriate alloys containing nickel. Send details of your metal problems for our suggestions. Write us today.

NICKEL CAST IRONS combine improved strength along with other desired properties such as resistance to heat and wear. The many types available allow you to specify thermal expansion, pressure tightness, machinability and the like. Typical of nickel iron performance, these melting pots cast by C. H. Milles Foundry Company, Chicago, Ill., for use at elevated temperatures, outlasted plain iron pots by 60%.

CAST NICKEL SILVER bottle cappers produced by Universal Casting Corp., of Chicago, Ill., are easy to cast or fabricate, machine and polish. This durable alloy combines corrosion resistance and lustrous beauty. Ease of fabrication and maintenance make it ideal to increase product acceptance and cut costs, too.

HERE'S NICKEL STEEL that withstands thousands of impacts per minute. It's a carburized grade used in hammer, anvil and arbor of an Ingersoll-Rand Impactool, shown here, bolting up an assembly. From the many standard grades of nickel alloy steels available, you can readily choose one with the best combination of properties for both fabrication and end use.

EMBLEM OF INCO SERVICE

THE INTERNATIONAL NICKEL COMPANY, INC. 67 WALL STREET NEW YORK 5, N. Y.

Benjamin F. Fairless, chairman of the board of U. S. Steel Corp. believes that the American steel industry next year will surpass by five to ten million tons, 1954 production. This year's production is expected to reach 86 million tons.

Waspaloy, a nickel base heat resisting alloy used in blades for the Pratt & Whitney J-48 turbojet, a centrifugal flow engine, has been found to stand up better under stresses of high temperature operation than previously used blade materials. Waspaloy uses over 50 pct nickel as a base. Chromium, cobalt, molybdenum, and small amounts of titanium and aluminum are added for stability and hardness.

U. S. Dept. of Labor is making a nationwide survey of the $2 1/2 billion a year research and development program of U. S. industry. The study is being carried out by the Labor Dept.'s Bureau of Labor Statistics for the National Science Foundation. It is hoped that the interviews planned with more than 200 companies will bring to light any bottlenecks faced by industry in developing research activities.

Lukens Steel Co., Coatesville, Pa., is carrying out an extensive research program which seeks to obtain fundamental data on the physical behavior of clad steels at elevated temperatures. Initially, the project aims at determining the physical properties of stainless-clad steel (18-8 type) at temperatures up to 1000°F.

A selenium rectifier for high temperature operations has been developed by Fansteel Metallurgical Corp., North Chicago, Ill. Company officials say it is designed to operate at the same current loads and voltage at temperatures up to 212°F as it does at room temperature.

Pill-sized piece of a radioactive element, strontium 90, is the brain that makes possible a machine that controls thickness of cold rolled steel. Employed in a radioisotope thickness gage, the element can detect extremely slight variations in thickness and, by means of electrical signals, activate control equipment.

Dominion Foundries & Steel, Ltd. announced operation of a new oxygen steelmaking plant which it termed to be the most modern steelmaking plant on the North American continent. The vessels, the first on this continent, are the largest installed to date. The proportion of steel to be produced by this oxygen steelmaking method in the future is estimated at between 60 and 80 pct.

Atomic Energy Commission approval has been given to a proposal by Bethlehem Steel Co.'s shipbuilding div. to study the application of nuclear power to commercial ship propulsion. The company has been associated with the development of machinery for the submarines Nautilus and Sea Wolf.

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Manager, Hydro-Arc Furnace Dept.
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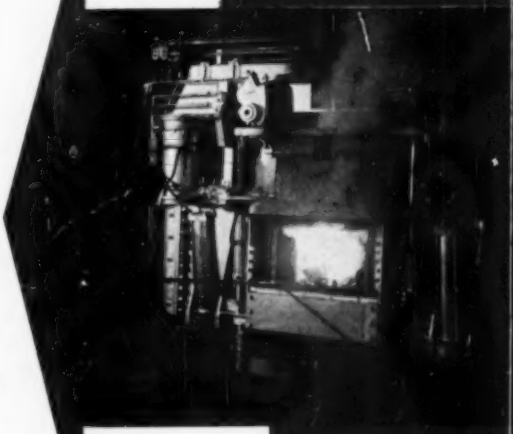
FLORIDA MACHINE & FOUNDRY CO.

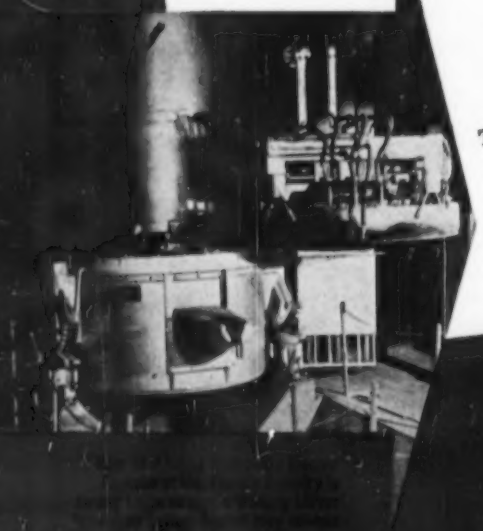
Thomas W. Peacock

Thomas W. Peacock
Assistant Superintendent

TWP/wb

ELECTRIC STEEL CASTINGS





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EFSC Attendance

1943	414	1949	585
1944	569	1950	620
1945	500	1951	752
1946	546	1952	823
1947	569	1953	749
1948	534	1954	?

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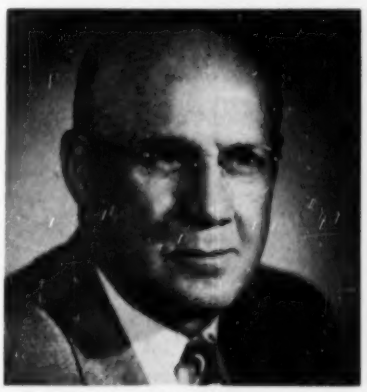
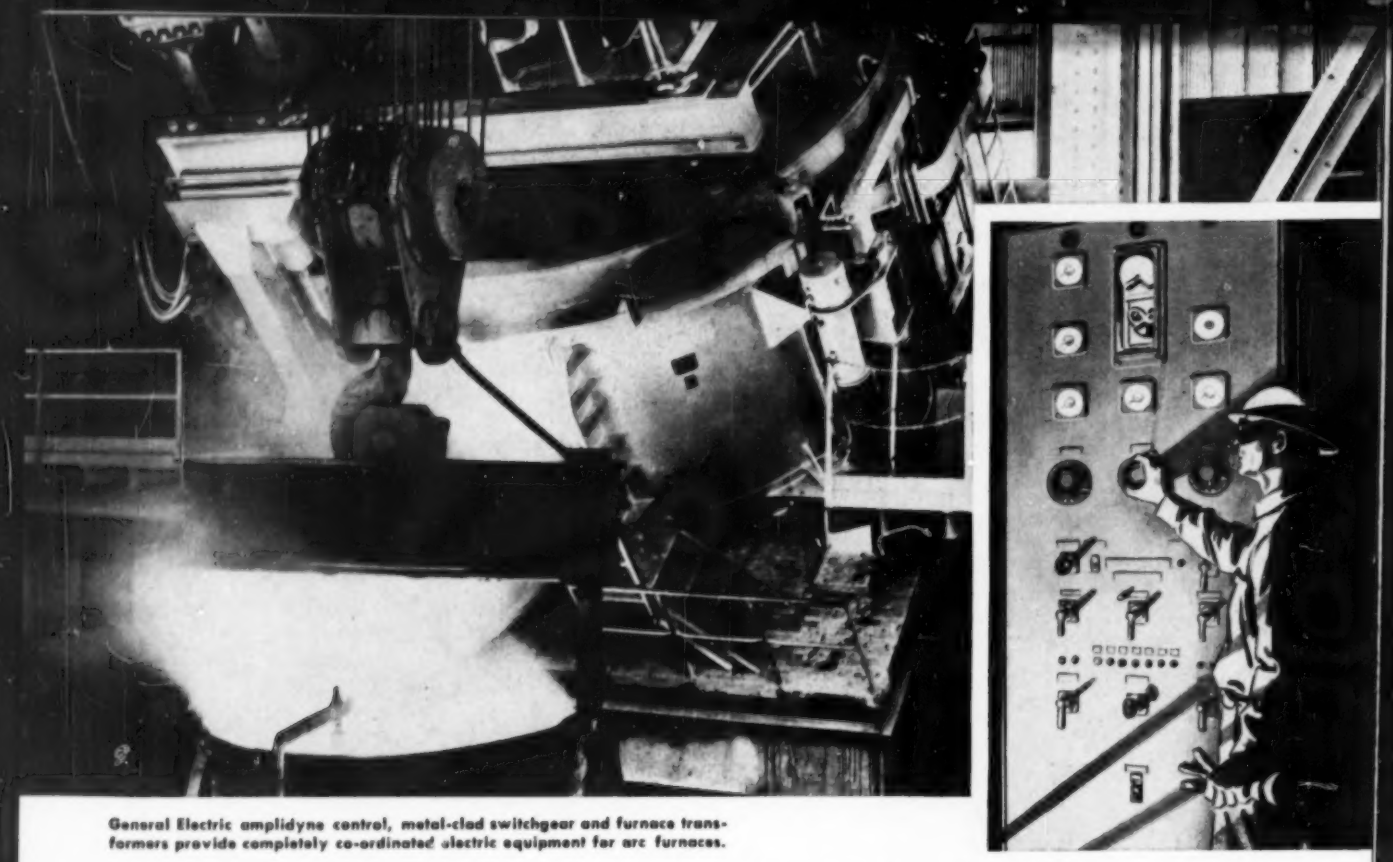
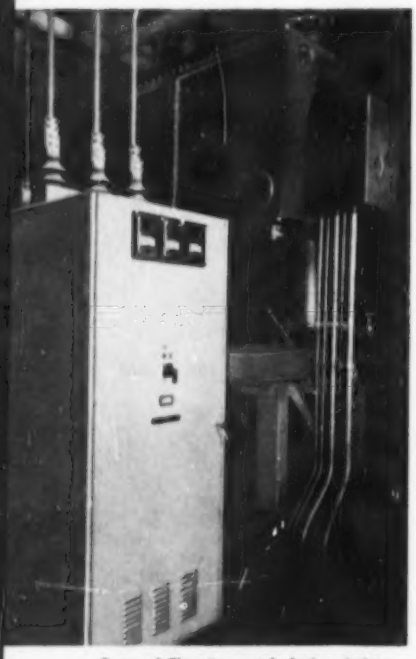


C. E. SIMS



General Electric amplidyne control, metal-clad switchgear and furnace transformers provide completely co-ordinated electric equipment for arc furnaces.

General Electric Regulators Provide Unexcelled Arc Furnace Performance



General Electric metal-clad switchgear is designed for the rugged duty of arc furnace switching.

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For more information about General Electric control for arc furnaces, see your G-E sales representative or write to General Electric Company, Schenectady 5, N. Y. 511-7

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Technical Program

Electric Furnace Steel Conference

William Penn Hotel, Pittsburgh

REGISTRATION

Wednesday, December 1—9:00 am to 9:00 pm—Sky Room, 17th Floor

Thursday and Friday, December 2 and 3—8:30 am to 5:00 pm—Sky Room, 17th Floor

WEDNESDAY, DECEMBER 1

Plant Trip: Leave by chartered buses from the Grant St. entrance of the William Penn Hotel at 1:00 pm and return not later than 5:00 pm. There is a choice of visiting the ingot producing shop of U. S. Steel Corp. at Duquesne or the steel casting shop of McConway & Torley Corp.

WEDNESDAY, DECEMBER 1

6:00 pm—Rooms 566-68-70

Electric Furnace Steel Committees Annual Dinner and Business Meeting.
(All Chairmen of technical sessions are invited to attend)

THURSDAY, DECEMBER 2

9:30 am to 9:45 am—Ballroom

GENERAL SESSION

Welcoming Remarks:
By C. F. Staley, Chairman, Committee for the Twelfth Annual Conference

Response:
By F. O. Lemmon, Chairman, Electric Furnace Steel Executive Committee—Announcements and Reports.

VACUUM MELTING

Chairmen:
W. M. Farnsworth, Assistant District Manager, Central Alloy and Chicago District, Republic Steel Corp., Massillon, Ohio
R. J. Wilcox, Technical Director, Michigan Steel Castings Co., Detroit, Mich.

1. **General Problems and Results of Refining Techniques**
   Paper by J. H. Moore, General Manager, Vacuum Metals Corp., Cambridge, Mass.
2. **Pertinent Items of Interest**
   A. Reason for Vacuum Melting
   B. Experience on Pilot Scale
   C. Improved Properties Through Vacuum Melting
   D. Experience in Mechanical Working
   Paper by R. K. McKechnie, D. W. Green, and W. F. Moore, Metallurgical Research Dept., General Electric Co., Schenectady, N. Y.
3. **Consumer Interest in Vacuum Melted Alloys**
   Preprinted paper by E. M. Mahla, Assistant Director, Mechanical Development Laboratory, E. I. du Pont de Nemours & Co., Wilmington, Del.
4. **European Vacuum Melting History and Practice**
   Preprinted paper by H. H. Scholefield, Metals Div., The Telegraph Construction & Maintenance Co., London, England

2:00 to 5:00 pm—Ballroom

REFRACTORY AND MASONRY

Chairmen:
D. L. Clark, Melting Superintendent, Simonds Saw & Steel Co., Lockport, N. Y.
L. W. Cashdollar, Metallurgical Engineer, Pittsburgh Metallurgical Co., Pittsburgh, Pa.

1. **The Evaluation of Electric Furnace Refractories Performance**
   Preprinted paper by M. P. Fedock, Ceramic Engineer, Republic Steel Corp., Canton, Ohio
2. **Bottom Materials and Bottom Maintenance**
   Paper by H. C. Bigge, Superintendent Tool Steel and No. 1 O.H. Dept., Bethlehem Steel Co., Bethlehem, Pa.
   Paper by A. C. Ogan, Superintendent Electric Furnace Dept., U. S. Steel Corp., Duquesne, Pa.
   Discussion by J. A. Voss, Jr., District Sales Manager, Kaiser Aluminum & Chemical Co., Akron, Ohio
   Discussion by C. R. Heilig, Sales Engineer, Basic Refractories Inc., Cleveland, Ohio
3. **Side Walls and Side Wall Maintenance**
   Paper by J. E. Harrod, Superintendent No. 2 Electric Furnace Dept., U. S. Steel Corp., Chicago, Ill.
   Paper by R. J. McCurdy, Superintendent No. 2 Melt Shop, Republic Steel Corp., Chicago, Ill.
   Discussion by Thomas Hand, Sales Development Engineer, Corhart Refractories Co., Louisville, Ky.
   Discussion by R. S. Moore, Assistant General Sales Manager, Harbison-Walker Refractories Co., Pittsburgh, Pa.
4. **Roof Construction and Maintenance**
   Paper by K. L. Grant, Open Hearth Superintendent, Bethlehem Pacific Coast Steel Co., Los Angeles, Calif.
   Discussion by E. L. Bohn, Vice President, Mullite Refractories Co., Shelton, Conn.
   Discussion by Addison Maupin, Assistant to President, Chas. Taylor Sons Co., Cincinnati, Ohio
5. **The Effect of Intermittent Operations on Electric Furnace Refractories**
   Paper by A. K. Blough, Superintendent No. 2 and No. 3 Melt Shops, Republic Steel Corp, Canton, Ohio
   Preprinted paper by R. P. Hill, Electric Furnace Superintendent, Sharon Steel Corp., Lowellville, Ohio
   Discussion by C. E. Grigsby, Service Research Engineer, General Refractories Co., Philadelphia, Pa.
   Discussion by Karl Sandy, Division Foreman, Mechanical Dept., Universal Cyclops Steel Corp., Bridgeville, Pa.

ANNUAL DINNER

THURSDAY, DECEMBER 2

6:30 pm—Urban Room, Reception and Cocktail Party for dinner guests

7:00 pm—Ballroom, Dinner

Toastmaster: D. R. Loughrey, Staff Industrial Engineer—Steelmaking, Jones & Laughlin Steel Corp.

Speaker: Pressly H. McCance, President, Duquesne Light Co., Pittsburgh, Pa.

Subject: Power and Steel

2:00 to 5:00 pm—Urban Room

METALLURGICAL SESSION ON MELTING

Chairmen:

C. G. Mickelson, Assistant Research Director, American Steel Foundries, Indiana Harbor Works, E. Chicago, Ind.

V. E. Zang, Vice President, Unitcast Corp., Toledo, Ohio

1. **Basic Electric Melting Practice for Quality Steel**
   Preprinted paper by A. F. Gross, Metallurgist, Ohio Steel Foundry Co., Springfield, Ohio
   Discussion by H. A. Young, Crane Co., Chicago, Ill.
   Discussion by L. L. Whitney, Assistant to Works Manager, American Steel Foundries, Indiana Harbor Works, E. Chicago, Ind.
2. **Acid Electric Steel Making Practice**
   Preprinted paper by C. C. Wissmann, Metallurgist, Los Angeles Steel Castings Co., Los Angeles, Calif.
   Discussion by J. Fuqua, Metallurgist, Calumet Steel Castings Corp, Hammond, Ind.
3. **High Alloy Steel Melting in Basic Arc Furnace**
   Preprinted paper by H. C. Templeton, Lebanon Steel Foundries, Lebanon, Pa.
   Discussion by W. C. Williams, Melting Superintendent, Waukesha Foundry Co., Waukesha, Wis.
4. **Melting Practice Comparison for High Alloys**
   Preprinted paper by P. R. Gouwens, Metallurgist, The Fahralloy Co., Harvey, Ill.
   Discussion by John McBroom, Stainless Foundry & Engineering Co., Milwaukee, Wis.

FRIDAY, DECEMBER 3

9:30 am to 12:30 pm—Ballroom

QUALITY

Chairmen:

G. G. Zipf, Superintendent Steel Plant, Babcock & Wilcox Co., Tubular Products Div., Beaver Falls, Pa.

R. B. Shaw, Manager, Electric Furnace Melting, Allegheny Ludlum Steel Corp., Brackenridge, Pa.

1. **The Fluorescent X-ray Spectrometer, A New Tool for Rapid Analysis Determinations**
   Paper by Howard Pickett, Industrial Products Section, General Electric Co., Milwaukee, Wis.
   Preprinted paper by M. F. Hasler, Technical Director, and J. B. Rittenhouse, Analytical Systems Engineer, Applied Research Laboratories, Glendale, Calif.
   Preprinted paper by D. C. Miller, Head of Engineering, North American Phillips Co., Mt. Vernon, N. Y.
2. **Use of Reusable Insulated Low Volume C&D Hot Tops for Yield Improvement**
   Preprinted paper by J. C. Carpenter, Metallurgical Engineer, Ferro Engineering Co., Cleveland, Ohio
3. **New Approach to Exothermic Hot Toppings of Ingots**
   Paper by Michael Bock, Vice President, Exothermic Metallurgical Products Co., Conneaut, Ohio
   Discussion by H. M. Evers, Assistant Electric Furnace Superintendent, Universal Cyclops Steel Corp., Bridgeville, Pa.
   Discussion by B. M. Shields, Chief Process Metallurgist, Duquesne Works, U. S. Steel Corp., Duquesne, Pa.
   Discussion by A. L. Lehman, Metallurgical Supervisor, Bethlehem Steel Co., Bethlehem, Pa.
4. **Addition of Special Minor Elements to Electric Furnace Heats**
   Preprinted paper by A. J. Scheid, Jr., Vice President in Charge of Metallurgy, and W. J. Mathews, Superintendent of Melting, Columbia Tool Steel Co., Chicago Heights, Ill.
   Paper by C. W. Sherman, Director of Research, Latrobe Steel Co., Latrobe, Pa.

9:30 am to 12:30 pm—Urban Room

OPERATIONS IN FOUNDRY MELTING

Chairmen:

C. B. Williams, Works Manager, Massillon Steel Castings Co., Massillon, Ohio

C. W. Vokac, Manager, Hydro-Arc Div., Whiting Corp., Harvey, Ill.

1. **The Use of High Alumina Brick in the Single Slag Basic Practice**
   Paper by V. E. Belusko, Metallurgist, Electric Steel Foundry, Portland, Ore.
2. **Operational Experience With a New Type Control**
   Preprinted paper by Joseph Seymour, Metallurgist, Kensington Steel Co., Kensington, Ill.
3. **Foundry Ladle Practice for Acid Steel**
   A. Motion Picture *Quarrying Operation for Gannister Production* presented by: S. E. McGinty, Sales Manager, Firegan Sales Co., Chicago, Ill.
   B. Paper by Wilfred Luvisi, Melting Superintendent, Bonney-Floyd Co., Columbus, Ohio
   C. Paper by G. R. McDaniel, Chief Melter, and R. S. Haught, Chief Metallurgist, Sawbrook Steel Castings Co., Lockland, Ohio
4. **Foundry Ladle Practice for Basic Steel**
   A. Preprinted paper by K. T. Apgar, Melting Superintendent, Taylor-Wharton Iron & Steel Co., Highbridge, N. J.
   B. Preprinted paper by R. H. Lewis, Chief Metallurgist, Buffalo Foundry, Allegheny-Ludlum Steel Corp., Buffalo, N. Y.

2:00 to 5:00 pm—Ballroom

INGOT BULL SESSION

Chairmen:

Gerhard Derge, Jones & Laughlin Professor of Metallurgy, Carnegie Institute of Technology, Pittsburgh, Pa.

T. V. Wainwright, Engineer, Research Dept., Bethlehem Steel Co., Bethlehem, Pa.

1. **Summary and Appraisal of Significant Features of the Conference in Relation to Immediate Operations.**
2. **Expert Discussion of Special Problems not Included in the Formal Program.**

Members are urged to submit written questions in advance. Send them to the secretary or session chairmen.

2:00 to 5:00 pm—Urban Room

CASTINGS BULL SESSION

Chairmen:

Charles Locke, Works Manager, West Michigan Steel Foundry Co., Muskegon, Mich.

J. B. Caine, Consultant, 5 Linden Dr., Cincinnati, Ohio

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Vacuum Melting Improves Alloy Properties And Workability

by R. K. McKechnie, D. W. Green and W. F. Moore

EXPERIENCE in the processing of more than 100,000 lb of alloys by vacuum melting has resulted in considerably higher and more uniform ingot-to-bar yields than conventional melting methods. Although melting experience at GE includes both iron and cobalt base alloys, operating experience described will deal with the nickel base alloy, M 252, whose nominal composition is given in Table I. Materials used in the pilot plant operation are all commercially available and are normally used without any pretreatment before melting in vacuum.

Vacuum melting equipment in the GE pilot plant can be considered as an induction furnace mounted inside a double walled, water cooled stainless steel tank connected to a pumping system. The two furnaces used are powered by a 200 kw, 1920 cycle motor generator set and operate with coil voltages as high as 400 v on the coil.

Pumping is provided on the roughing line by a 200 cfm mechanical pump in the range from atmospheric to 200 microns (0.2 mm) pressure. A separate high vacuum line contains two booster diffusion pumps in series with a speed of 1000 cfm at 10 microns. Gaging consists of four thermocouple gages located one each at the tanks and in the pumping lines. An Alphatron gage (10 mm-1 micron range) and a McLeod gage (5 mm-0.1 micron range) are used interchangeably on the two tanks.

Auxiliary equipment and devices include mechanisms for making additions to the melt, bridge breaking bars, gas inlet tubes, remote controlled pouring, thermocouple access tube, and sampling devices. The 50-lb furnace is equipped with a table for centrifugal casting. Preformed or rammed crucibles which have been used in both units include magnesia, zirconia, zircon, magnesia-alumina mixtures, and graphite.

In contrast to conventional melting installations for which direct access to the melt is available at all times during processing, provisions must be made for carrying out all operations without breaking the vacuum in these batch type units.

Operation

A typical operating cycle for the 500-lb furnace is shown in Fig. 1. The bulk charge is first loaded in the crucible in air and additions are placed in the charging devices. Ingot molds which have been warmed by a gas torch are placed in pouring position, the furnace top is closed, and rough pumping is begun. Power is applied during this part of the cycle to aid in outgassing the charge before melting in the heat. At melt down a pressure rise occurs immediately as dissolved gases are released, adsorbed gases are driven off hot refractory, and volatile materials from the charge are vaporized. A carbon boil occurs simultaneously. If a second bulk addition is made, a burst of gas like the original melting in will be obtained. When a hydrogen treatment is used, it is the next step in the process with the gas flow impinged on the melt at 100 to 150 cu ft per hr and pressures less than 40 mm. This treatment aids in the removal of nitrogen, reduces the bath oxygen level,[1-2] and cleans up any slaggy material formed during the melt down of raw ma-

R. K. McKECHNIE is Research Associate and D. W. GREEN and W. F. MOORE are Research Assistants, Metallurgy Research Dept., General Electric Co., Schenectady, N. Y. This paper will be presented at the AIME Electric Furnace Steel Conference, Pittsburgh, Dec. 1 to 3, 1954.

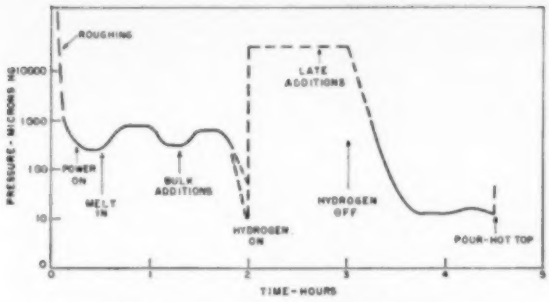

Fig. 1—Typical operating cycle in a 500-lb furnace shows the steps in melting a heat under vacuum.

terials. Such a treatment has been shown to reduce the bath oxygen by 20 pct in the case of M 252.[2]

Gases and water vapor are continuously pumped out with the hydrogen during this phase of the cycle. Reactive additions such as aluminum, boron, carbon, calcium, manganese, magnesium, columbium, silicon, tantalum, titanium, and zirconium are made during the last 10 min of this treatment. Hydrogen flow and power are turned off after the additions have dissolved and the melt is then pumped until the leak-up rate is satisfactory. (The leak-up rate is determined by blanking off the pumping system from the tank and measuring the rate of rise of pressure in microns multiplied by the volume of the system being measured.) The charge is then brought to pouring temperature as rapidly as possible and is cast into the mold at pressures generally below 50 microns. A pressure rise occurs on pouring as refractories not previously heated to the molten metal temperature are touched. Electric hot topping provides a sound ingot with little metal in the hot top section. Fig. 2 shows the ingot mold and hot-top assembly and the typical appearance of several 450-lb ingots.

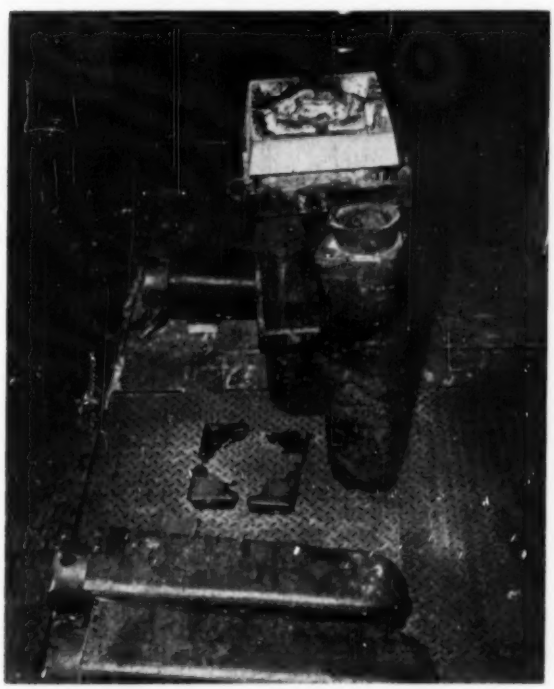

Fig. 2—Ingot, mold, and hot top assembly are shown with several 450-lb ingots.

Mechanical Working

The difficulty in cogging and rolling alloy grades high in titanium and aluminum is well known from air melting experience. Experience may vary from poor recovery due to excessive gouging and grinding due to face and corner cracks to almost total loss of the ingot from gross break up during these operations. It is generally agreed that this behavior is due to a metallurgically dirty product, and it appears that nitrogen is the prime offender.

Oxygen and nitrogen are not present in the furnace atmosphere (neglecting the outgassing of the charge) and consequently the ingots produced are

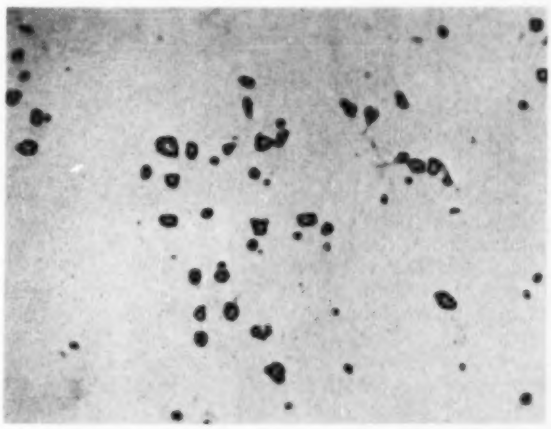

Fig. 3—As cast microstructure of a 30-lb vacuum melted M 252 ingot is shown unetched and magnified 500X.

clean as shown in Figs. 3 and 4. A typical analysis for oxygen and nitrogen in vacuum melted M 252 is: oxygen—0.0005 to 0.001, nitrogen—0.002 to 0.003. Fig. 3 shows the as cast microstructure of a 30-lb vacuum melted M 252 ingot and Fig. 4 the macrostructure of the same ingot. The second phase material consists of M_6C carbides. The structures of 450-lb ingots are essentially the same.

An improved workability of vacuum melted heats is evident from the first hammer blow in the cogging operation. On the nominal 7 in. ingots described, the working schedule was as follows:

A—Condition ingot by overall grind to remove surface defects.

B—Hammer forge to 2½ in. sq rolling billets. Spot grind.

C—Rough roll to 1¾ in. sq billets. Spot grind.

D—Finish roll and centerless grind to size.

A summary of the centerless ground bar/as cast ingot yield for 60 M 252 ingots is shown in Fig. 5. One ingot which developed a deep pipe due to a poor

Table I. Nominal Composition of Nickel Base Alloy, M 252

Element	Pct	Element	Pct
Fe	5 max.	Ti	2.5
Cr	19	Al	1.0
Co	10	C	0.15
Ni	Bal	Mn	0.75
Mo	10	Si	0.50
W	—	V	—
Cb	—		

Fig. 4—As cast macrostructure of a 30-lb vacuum melted M 252 ingot is shown. X marks the center of the ingot.

pour was omitted from the summary. The range of yields is from 52 to 72 pct, and the average yield is 63 pct. These uniformly high yields can be attributed directly to low gouging losses and to fewer defects to be removed in billet conditioning. Since the 60 ingots represent 60 different heats melted during a four month period, the values take on added significance. Additional yield data of interest include the processing of 100 pct scrap charges which were melted to consider the consumption of scrap under extreme conditions. On a smaller group of heats, the range fell to 47 to 61 pct and the average decreased to approximately 57 pct.

Because vacuum melted M 252 does not require the residual manganese and silicon specified in the conventionally melted grade, yield studies have been made on manganese and silicon-free material. These data suggest that an average yield of 70 pct can be expected for M 252 which does not contain manganese or silicon.

Another measure of workability is found in the performance of bar stock in its end use, in this case aircraft gas turbine buckets. The most severe operation in the forging of these shapes is in the initial upsetting of slugs for the forming operation. Twenty-nine of the 60 heats were subject to the same upset from ¾ in. centerless ground bar. An arbitrary defect frequency rating was established which rejects from the normal process flow any material with a rating above 60. From the frequency bar graph in Fig. 6 it can be seen that the piling-up tendency is toward the lower numbers, i.e., heats with good upsettability. Only two of the 29 heats would require non-standard processing according to this criterion.[3]

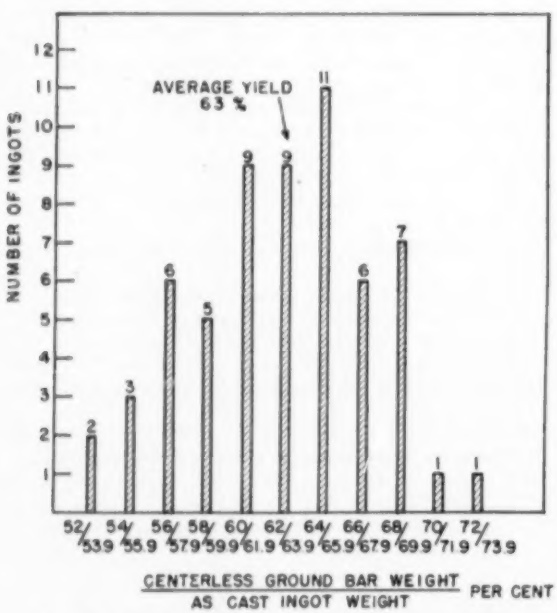

Fig. 5—Frequency diagram for 60 ingot yields of M 252 shows the reproducibility of quality heats in vacuum melting.

Improved Properties

Improvements resulting from vacuum melting can be considered under two categories:

A—Materials which can be processed by conventional methods.

B—Materials which require vacuum melting due to composition.

In the first case, the improvements can be ascribed to the vacuum melting process *per se*, while in the second case the improvements are largely due to new alloy compositions which can be produced successfully by the vacuum melting process.

Fig. 7 shows the average tensile elongation of conventionally melted, vacuum melted, and vacuum melted manganese and silicon-free M 252 in the temperature range 1200°F to 1500°F. Vacuum melting alone produces a significantly more ductile alloy and that the material not containing manganese and silicon has almost three times the elongation at the critical 1350°F temperature as the vacuum melted alloy with these elements. From the ductility standpoint, therefore, the elimination of the residuals results in a different alloy.

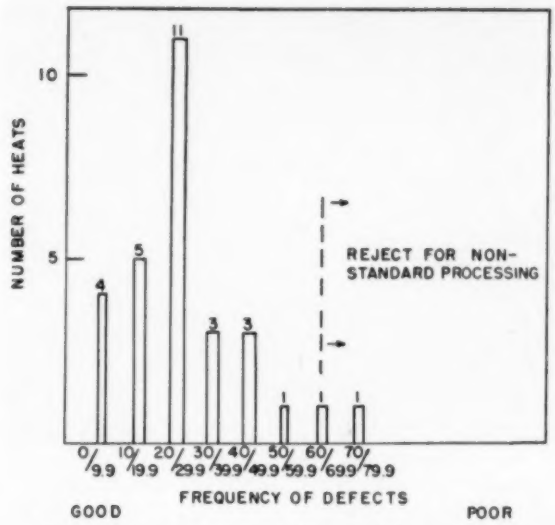

Fig. 6—Frequency diagram of the upset defect rating of 29 heats of M 252 shows that only two of the 29 heats required non-standard processing.

Table II. Nominal Composition of Cobalt Base Alloy, 1570

Element	Pct	Element	Pct
Fe	2.5 max.	Ti	4.2
Cr	20.0	Al	—
Co	Bal	C	0.20
Ni	29.0	Mn	—
Mo	—	Si	—
W	7.0	V	—
Cb	—		

Considering the strength in stress-rupture, Fig. 8 compares the average rupture stress at 1000 hr and 1500°F for conventionally melted and vacuum melted M 252 of the same analysis and alloy 1570. Table II gives the analysis for the cobalt base alloy, 1570. The vacuum melted material shows a 10 pct improvement in stress compared to the conventionally melted material. More important, alloy 1570 which uses an amount of titanium requiring vacuum melting shows an increase of about 30 pct.

The more reproducible and uniform nature of the properties of vacuum melted alloys will be presented in greater detail in another paper.[4] Other wrought alloys of this general type with still better stress-rupture characteristics are currently being evaluated in the General Electric Co.'s Research Laboratory.

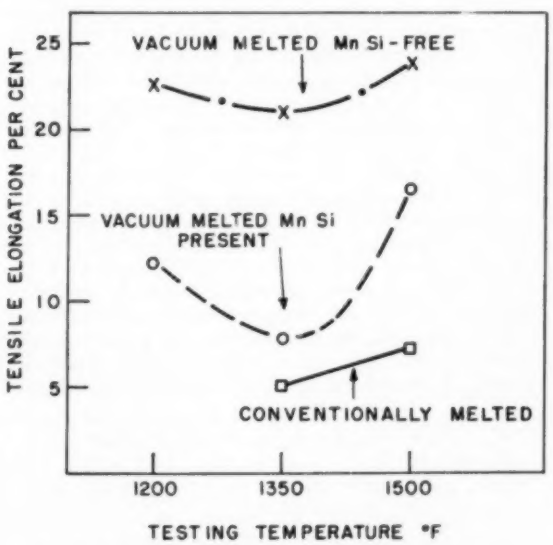

Fig. 7—Comparison of elevated temperature tensile elongation between vacuum melted and conventionally melted M 252 shows increase in ductility from vacuum melting.

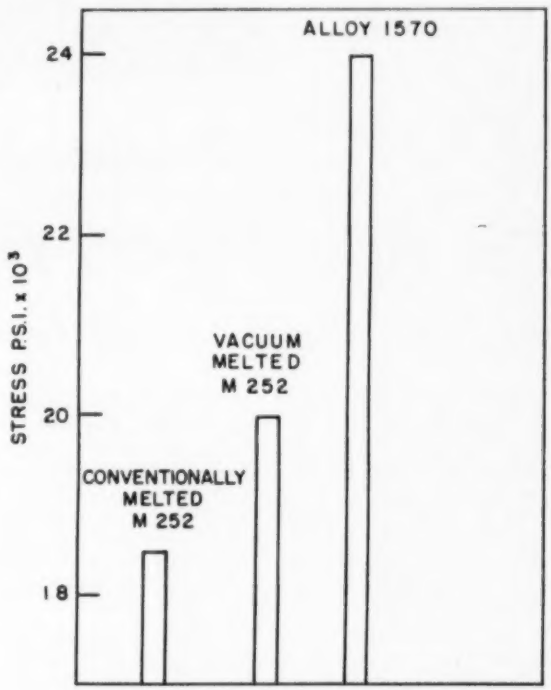

Fig. 8—Comparison of stress for rupture in 1000 hr at 1500°F is shown between vacuum melted M 252, alloy 1570, and conventionally melted M 252.

Acknowledgments

It is a pleasure to recognize the contributions of T. A. Prater, R. M. Parke, and J. H. Hollomon, GE Research Laboratory and the support of I. L. Wallace and W. E. Jones, Carboloy Dept., General Electric Co. The enthusiasm of a former co-worker in the research laboratory, J. D. Nisbet, now Universal-Cyclops Steel Co., is gratefully acknowledged. The suggestions and close liaison afforded by C. F. Sawyer and G. A. Roberts, Vanadium-Alloys Steel Co. were most valuable.

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[2] W. F. Moore and R. K. McKechnie: Deoxidation of Vacuum Melted M 252. Vacuum Melting Symposium. Electrochemical Society, Oct. 6 to 7, 1954.

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[4] D. W. Green, R. K. McKechnie, and T. A. Prater, data to be published.

Melter checks temperature with an optical pyrometer during a melt in a 600-lb furnace.

Promises and Problems Posed by Vacuum Melting

by James H. Moore

LIKE many technological developments, vacuum melting is not a new concept, but it has had to wait for the development of both need and equipment to emerge from the status of a laboratory tool to that of a production process. It became a possibility when high temperature heating within an enclosed chamber could be accomplished by high frequency induction, and a practicality with the recent development of the high capacity high vacuum pumping system.

There are many cases of limitation in equipment performance caused by the inadequacy of metallic materials. In jet aircraft there is constant pressure for higher temperature operation and increased service life which seem to be hitting a ceiling in the available air melted alloys. The increase in size and power of aircraft and land vehicles alike requires higher loading of components in engines and structures, and a wider range in operating temperatures. The requirements for automatic control call for electronic units which often depend, in the final analysis, upon the behavior of certain metallic components. These metals must exhibit a consistency in properties and a freedom from tramp impurities and gases which points toward vacuum melting for such precise control.

Fig. 1 and Table I illustrate the relative impact properties of air and vacuum melted ferritic stainless steels. The latter exhibit no transition from ductile to brittle failure until about −50°C, while the ordinary alloys are brittle not only at room temperature but also appreciably above it. The improvement in the room temperature impact strength of the high chromium alloy is nearly 50 to 1. Fig. 2 gives the high temperature properties of a vacuum melted nickel base alloy.

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Fig. 3 gives fatigue and service test data for two steels. In the rotating beam test bars, the shading indicates roughly the spread in the laboratory test results. This idealized test shows a much greater effect than that demonstrated by ball life tests, but the latter show a very significant improvement through vacuum melting. Vacuum melted 4340 steel has exhibited not only increased fatigue strength but also decreased anisotropy in fatigue properties.

Property Improvements

The improvements being achieved by vacuum melting appear to fall into three basic categories: A—those which are the result of increased freedom from inclusions, B—those achieved by the elimination of gaseous or gas-metal compound precipitation, and C—those resulting from better composition control. In most alloys all three effects may be obtained, although their relative importance varies.

In a precipitation-hardened high temperature alloy, for instance, the reduction in inclusions improves hot working, the reduction of oxygen and nitrogen contents is a probable cause for improved intergranular strength at elevated temperatures, and a closer control over titanium and aluminum hardening additions gives more uniform heat treatment response. In an alloy steel, increased cleanliness will improve working characteristics, and properties such as fatigue strength and tensile ductility. Reduction in gas content which would otherwise result in pre-

Table I. Room Temperature Charpy Strength

Alloy	Ft Lb
Ferrovac*	220 +
Air Melt	8
Ferrovac*	144
Air Melt	3

* Trade mark, Vacuum Metals Corp.

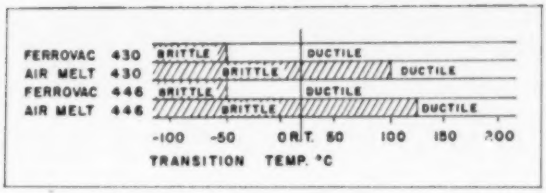


Fig. 1—Impact properties of air and vacuum melted ferritic stainless steels are compared.

cipitation, can greatly increase the strength and ductility at high strain rates.

These advantages of vacuum melting are the plus side of the ledger; on the other side there are problems, some of which are economic, and others, technical. It is obvious that, with a piece of equipment having a higher capital investment, a capacity much smaller than that of the arc furnace, a somewhat larger labor and maintenance requirement, and a longer operating cycle, an appreciable price premium must be expected. Even at the high prices made necessary by pilot plant operations vacuum melted products have found application for very critical uses. The extension of their use to applications in which they replace substantial amounts of conventional materials will depend upon a favorable balance of premium behavior against premium price. To the vacuum melter, this will mean constant pressure to decrease costs without sacrificing quality, and a concentration on the alloys which have most potential for such a balance.

A number of factors affect the economic picture. A balance must be struck, for instance, between the saving in material cost from using a more impure starting material and the increased cycle that may be necessary to accomplish the refining. In degassing or deoxidizing with gaseous products, the mass transfer of gas as a function of pressure is very important, because a necessity for a large transfer at very low pressure would lead to a long cycle. Another economic problem is the definition of quality control specifications. In some cases, a purification past a certain point may materially increase manufacturing cost with only a marginal increase in properties.

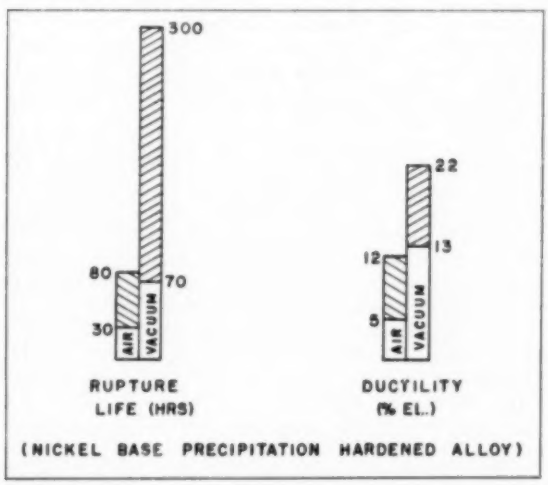


Fig. 2—Rupture life and ductility for air and vacuum melted specimens of a nickel base precipitation alloy demonstrate the desirable properties obtainable by the vacuum process.

Future Considerations

Many technical problems are being attacked and will require future attention. The role of the furnace refractories is more important than in air melting. Refractories are reduced by the bath more easily in vacuum, and when very high purity products are being made, contamination from this source may be significant. Because of the necessity for remote operation, it is more difficult to repair the refractory components than in open melting, and the effect of the heating and cooling cycles may be more drastic. Basic linings or crucible have been most used for vacuum induction melting, and their tendency to develop thermal cracks creates a problem in lining life which is both an economic and a safety factor.

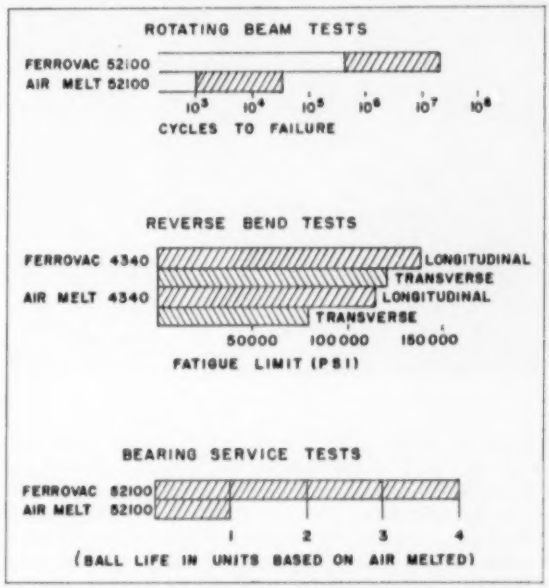
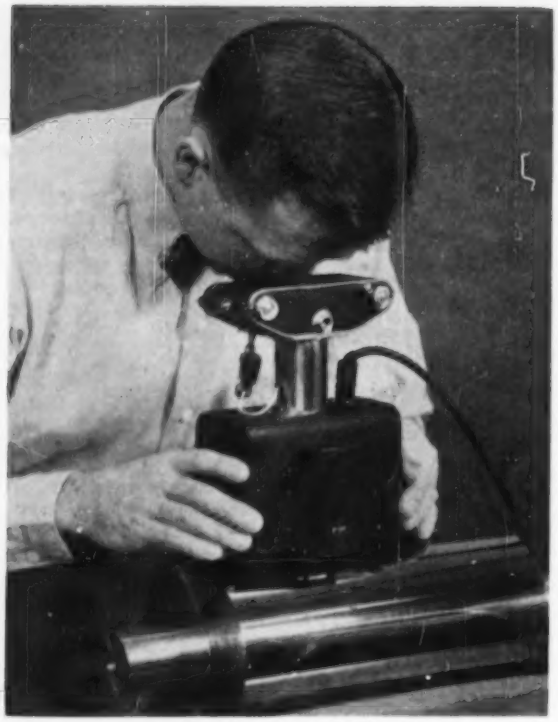


Fig. 3—Fatigue and service test data are shown for two steels. Ferrovac is a trade mark of Vacuum Metals Corp.

It is a matter of vital interest to vacuum melters that they understand the problem caused by water leakage in a vacuum furnace operation, whether it be caused by a lining break-out or a leak from some source such as the water cooled induction coil. When water vapor contacts molten steel, hydrogen is formed. If oxygen is then allowed to enter the tank, the explosive hydrogen-oxygen chain reaction becomes possible because of the confinement of the mixture. Safety regulations and procedures to meet this problem are mandatory.

The application of vacuum to melting gives the process metallurgist control over the powerful variable of gas pressure and to a certain extent that of gas composition. It can be nullified by lack of consideration of other variables in processing. A melt which has been vacuum purified can become badly contaminated during tapping by exogenous inclusions coming from refractory attack and disintegration. The beneficial effects of very low gas and nonmetallic content can be realized only if all other factors affecting quality are under control, particularly chemical homogeneity and soundness. In short, the vacuum metallurgist is first a metallurgist and second a vacuum expert who applies knowledge of equilibria and kinetics as affected by pressure.



Bright Future Predicted For Vacuum Melted Metals In Chemical Industry

by E. M. Mahla

PROCESSING procedures that can improve the useful property range of steels will be fostered by the chemical industry in order to extend the list of useful products or reduce the cost of existing products. Vacuum melting appears to offer promise by improving many properties of steels and is thus of current interest to the industry.

The benefit that is expected from these steels can be exemplified currently by three principal fields of interest. All appear to be associated with the essentially inclusion-free surface of these materials. They are: A—transverse fatigue properties; B—high quality surface for film casting wheels; and C—corrosion resistance.

It is difficult to say how much these fields of interest may be extended. There are indications that the transformation kinetics of vacuum melted SAE 4340 differ significantly from commercial 4340. It may be that wear resistance, coefficient of friction, or even the quality of welds in some specialized materials can be usefully modified by vacuum melting.

How do transverse fatigue properties enter into the picture? An important line of manufacture at du Pont and the chemical industry in general relates to products synthesized under very high pressures—up to 30,000 psi. The equipment involved in generating such pressures and handling process streams is subjected to severe fluctuating internal stresses. The fatigue failure of such equipment, particularly pump and compressor cylinders, constitutes a high maintenance cost item. The cylinders are expensive, each one representing fabricating and machining costs many times the cost of the base steel, and labor involved in replacement is great. Plants using such equipment are in continuous operation. Down time for replacement involves the loss of valuable production. When an interrupted process is restarted a certain amount of off quality product is unavoidably produced. Considering these factors, it is easy to see that the failure of a single pump or compressor cylinder containing of $200 to $1000 worth of steel can be expensive.

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Fatigue Investigation

In a series of fatigue researches[1] carried out in du Pont laboratories, the following was shown:

A—In laboratory fatigue tests on standard aircraft quality AISI 4340 steels failure of specimens cut transverse to the forging direction always nucleate at a surface nonmetallic inclusion.

B—Transverse endurance limit, determined statistically by the staircase method on the same commercial AISI 4340 steel, was only 65 pct of the longitudinal endurance limit.

C—The transverse endurance limit of vacuum melted AISI 4340, having the same chemistry, the same forging reduction, and heat treated to the same strength level was 90 pct of the longitudinal endurance limit. In actual numbers, at a hardness of 28 to 29 Rc, the commercial steel had a transverse endurance limit of 0 to 83,000 psi, while the vacuum melted steel had a limit of 0 to 120,000 psi.

This difference in endurance limit of 37,000 psi is significant, particularly when it can mean the difference between infinite life and failure for an operating piece of equipment.

If the laboratory data can be substantiated in actual service tests the chemical industry would be justified in going to vacuum melted steels for high pressure equipment. It would be surprising if similar justification could not be obtained in many other applications for high strength steel where fatigue failure is a problem.

Consider the case of high quality surfaces for film casting wheels. Film is continuously cast on large revolving wheels, stripped off, further processed, slit to useful widths, and wrapped up prior to use. Since film materials are usually liquid when cast, they rather faithfully reproduce the surface quality of the casting wheel. Any defects on the film decrease salability of the product.

Du Pont Co.'s extensive research on wheel finishing procedures continuously strives toward perfection in wheel surfaces. As an outgrowth of these studies, the laboratories developed techniques for surface finishing where the limiting defects in metal surfaces have become the nonmetallic inclusions in the metals used for wheel fabrication. It has been shown that vacuum melted, corrosion resistant steel, essentially free of nonmetallic inclusions, is a superior material for casting wheels. Unfortunately, large forgings for 40 in. diam rolls on which to cast 36 in. film, or wider is still unavailable. It isn't difficult to imagine a large market for rolls having superior surface finish and extreme accuracy. It should be possible to produce superior foils, 0.005 to 0.0001 in. thick of a fairly wide variety of metals and plastics, if roll surfaces were sufficiently good.

Cylindrical surfaces are not the only considerations from the standpoint of surface finish. Many polymeric materials are extruded from flat dies with spaced lips. The manufacture of such extrusion dies is a tedious and costly process, requiring extreme geometric precision and superior surface finish. Imagine how frustrating it is to spend hundreds of man-hours finishing such die lips only to uncover gross nonmetallic defects or even microporosity in the last few steps. In a case like this, how important is a cost difference in the base material?

Corrosion Resistance

One of the most insidious forms of corrosion encountered in metal service is pit corrosion. In this type of corrosive action, localized areas on a metal surface are preferentially attacked. Corrosion may proceed until the entire cross-section of the part is perforated. Failure due to pitting is difficult to repair and usually results in discarding the equipment.

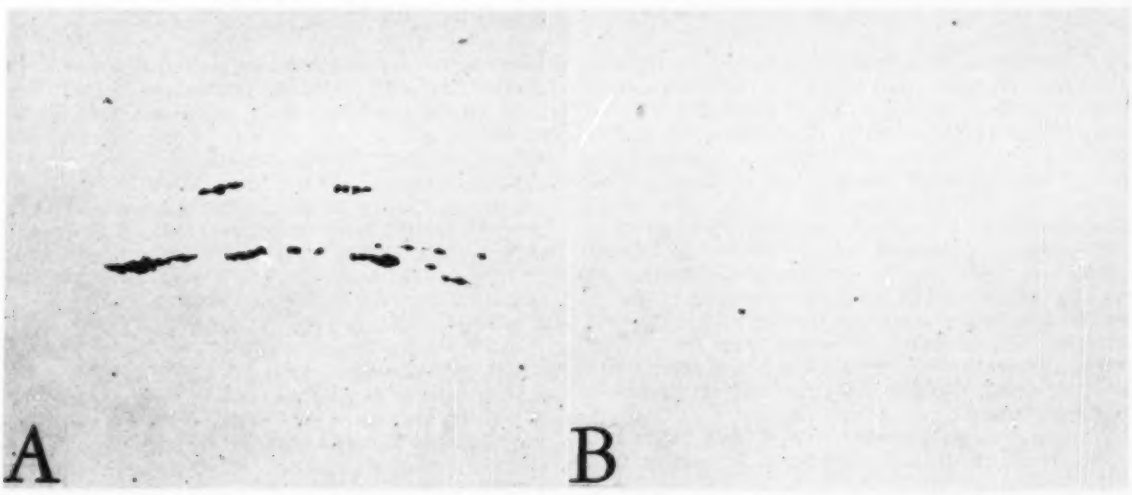
Laboratory research on pit corrosion, where the fine structure of stainless steels is studied relative to pit initiation, shows that nonmetallic inclusions located in grain boundaries are the most likely sites for the initiation of corrosion pits. Unfortunately, for this discussion of vacuum melted steels, these are not the only sites of pit nucleation. It must be concluded that metals fail by pit corrosion whether or not they contain nonmetallic inclusions in their surfaces. How much of an improvement inclusion free metals would afford relative to pit corrosion resistance, is unknown at this time. However, it is safe to expect metals to be measurably better under appropriately defined conditions. If a real tie can be established between surface nonmetallics and pit corrosion resistance, the interest of the chemical industry in vacuum melted stainless alloys will be great indeed.

As a major consumer of metals, the du Pont Co. has a real and active interest in vacuum melted steels, based on research already carried out in its laboratories. It is clear that under appropriate conditions the transverse endurance limit of inclusion free, vacuum melted steels is significantly superior to that of aircraft quality steels. Under all conditions, inclusion free steels make superior casting wheels and rolls to those made from standard materials. There is evidence that pit corrosion resistance may be measurably affected by surface nonmetallic conclusions.

All these facts, and many more open to speculation, such as heat treatment response and general corrosion resistance that could be affected by trace elements controlled by vacuum melting, make these steels of great current interest in research. Du Pont confidently expects to specify vacuum melting techniques for equipment as soon as the industry is able to supply it. It should not be construed that du Pont feels vacuum melting is the only way, or even the best way to obtain materials having low inclusion counts. Use of appropriate scavengers, coupled with proper processing control, may lead to the same goal by more economical means.

Reference

[1] J. T. Ransom: Effect of Inclusion on the Fatigue Strength of SAE 4340 Steel. *Trans.* ASM (1954) **46**, p. 1214.



Representative microstructures of typical nonmetallic inclusions for SAE 4340 in unetched specimens magnified 250X are shown in: A—commercial quality steel, and B—vacuum melted steel.

Temperature of electric furnace steel is measured in a well slagged spoon calculated to contain 585 ml with a normal metal content of 5 lb. A constant amount of metal in the spoon and an immersion depth of 1¼ in. are especially important.

Electric Furnace Melting Practices Compared for High Alloy Steel Production

by P. R. Gouwens

ACID dead melting, acid oxygen, and basic oxygen practices were compared in a Pittsburgh Lectromelt Type ST furnace. The furnace has a three phase direct arc using 41-in. diam electrodes and is powered by a 500 kva transformer. Regulation is accomplished through Amplidyne generators. The voltage taps available are: No. 1 Tap, 199v; No. 2 Tap, 164.5v; No. 3 Tap, 115v; and No. 4 Tap, 85v.

This type of furnace has a nominal rating of 1000 lb of metal per hr. Although this furnace is capable of producing much larger heats, foundry limitations have forced a normal charge weight of 1200 lb. The charging is accomplished by a drop bottom bucket. This installation has a close fitted hood mounted directly on it for exhausting fumes to the outside.

The castings poured vary from 0.03 lb to 2500 lb each. Most of these are made in extremely thin metal sections, and many are poured in green sand molds. About 90 pct of the production of Fahralloy is of the heat resistant grades with the major emphasis on the HH (25 pct Cr, 12 pct Ni) and HT (15 pct Cr, 35 pct Ni) grades.

In the melting practices to be described, the oxygen comes from a bank of cylinders. Oxygen pressure is adjusted for 60 psi gage pressure during the blow period. This is the pressure at the head end of 20 ft of oxygen hose. Oxygen is injected at 135 cfm with the injection at the slag-metal interface.

It is standard practice to blow all heats whose final carbon requirement is below 0.70 pct. Applied to the heat resistant HH and HT grades, melt down carbons of 0.08 and 1.00 pct respectively are the target. Heats are blown for 1¼ min by stopwatch ending with 0.40 and 0.50 pct carbon respectively. Oxygen consumption is about 7 cu ft per point ton, an efficiency in excess of 90 pct.

In 1952 an electronic immersion pyrometer using a Pt-Pt Rh thermocouple was put into use. Standard

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This paper will be presented at the AIME Electric Furnace Steel Conference, Pittsburgh, Dec. 1 to 3, 1954.

operating procedure calls for measurement of the temperature in a well slagged spoon, calculated to contain 585 ml. This volume is decreased by the slag coating so that the normal metal content is 5 lb. The spoon technique was dictated by the high temperatures in the furnace which in most heats is beyond the melting point of the thermocouple. This technique, as developed by Faist, is capable of introducing a uniform temperature drop so that readings are practical and economical.

Tapping temperatures vary, depending on the requirements of the foundry and the alloy composition, but for an average heat where 65 pct of the metal is shank-poured, the HT alloy would be tapped at approximately 2970°F and the HH alloy at approximately 3030°F using the spoon test. This corresponds to a true furnace temperature of 3195° and 3255F°, respectively.

Refractories

The refractories for both the acid dead melting and the acid oxygen injection processes were the same. The roof was made of either shaped super duty silica brick or of rammed monolithic super duty plastic refractories. The monolithic was preferred when the full furnace power had to go on immediately without the benefit of a long slow increase in temperature. But essentially both refractories were used with equal success.

These same roof refractories were tried on the basic practice with far less success, as can be seen from the tons per roof figure in Table I. The explanation for this is twofold. First, the basic slag is far more reflective of the heat from the arc than the acid slag. Secondly, no electrical adjustments have been made on the furnace since it was originally adjusted for peak performance on acid practice. As a result of the higher electrical conductivity of the more easily ionized basic slags, the arc is longer and radiates more heat to the roof. Furthermore, the slag volume is less so that a greater fraction of the arc length is exposed. The side walls, jambs, and arches were made from the super duty silica brick for the dead melting and the acid oxygen processes. The hearth was rammed in up to above the slag line using a mixture of crude sand, molasses, and water. This was also used for patching the slag line after every heat.

Table I. Refractory Life and Costs

Item	Acid Dead Melt	Acid Oxygen	Basic Oxygen
Heats per roof	635	452	102
Heats per lining	793	677	575
Tons per roof	323	255	59
Tons per lining	403	383	322
Roof cost per ton	$0.234	$0.302	$1.30
Lining cost per ton	$0.284	$0.299	$0.647

The basic lining consisted of a burned chrome magnesite brick for the side walls and bottom. Over the bottom brick a layer of 3 mesh chrome ore bonded with 10 pct periclase magnesite cement and water was hard rammed in place and subsequently dried and burned in place.

It is believed that the very high temperatures resulting from the oxygen blow are primarily responsible for the slightly lower life of the acid oxygen practice compared to acid dead melting, but the life of the roofs was also shortened by chipping away splatter from the electrode holes. This splatter caused a partial short circuiting of the electrodes and, therefore, poor response of the electrode regulator. As the roof became thinner, the chipping of the electrode holes without severely damaging the roof was almost impossible, so that once this condition developed the roof life was nearing its end.

Summary of Melting Practice Comparison for High Alloys

A—Acid, dead melting has inherent limitations on the type material charged into it which limits the ferroally additions.

B—In either acid dead melting or acid oxygen practice the oxidation loss of chrome and manganese is high.

C—Acid oxygen practice increases slightly the refractory and electrode costs but decreases power costs.

D—Cost of oxygen and other costs related to it, and increased chrome loss are more than balanced by savings from using low cost ferrochrome in the acid furnace.

E—Very careful control of temperature is mandatory for reasonable chrome recoveries when using oxygen in the acid furnace.

F—Cost of electrodes and power are reduced using a basic chromite furnace, but refractory costs are much higher.

G—Use of lower cost high silicon ferroalloys reduces costs of basic charge.

H—Excellent chrome recovery and improved manganese recovery substantially helps to make basic oxygen practice the preferred one from economic considerations.

I—So far as can be determined neither basic nor acid oxygen processes has a deleterious effect on physical properties of alloys.

J—Control of porosity is excellent when using oxygen but can be upset by over reduction of basic slag.

K—Metal fluidity is observed to be slightly superior in basic melting.

L—Slag fluidity presents a real problem to the foundry and needs more study.

M—More metal is tapped from the basic furnace with a given charge size than with either acid oxygen or acid dead melting.

Electrode Consumption and Power Use

Table II lists the electrode consumption for the three melting practices. The increased consumption with the use of oxygen is unusual, and is believed to be caused by the oxidation of the electrode during the injection period. The electrodes are raised about 2 ft during the blow which brings the penciled down part of the electrode up into the electrode gland. The pressure in the furnace and the draft from the exhaust rotoclone on the furnace hood causes a large volume of the high oxygen content furnace gases to sweep out through the glands past the hot electrodes resulting in their oxidation.

The variations in power consumption for two different alloys and for the entire production for the

Table II. Electrode Consumption

Practice	Lb per Ton
Acid dead melting	9.70
Acid oxygen	9.88
Basic oxygen	9.25

three melting conditions are given in Table III. The effect of increased temperature from oxygen is apparent and was increased even more in the basic practice. Further improvement is expected when the arc length is shortened causing a reduction in the heat losses to the roof and sidewalls.

Normally, an increase in power consumption might be expected when converting to basic practice because of the greater heat conductivity of the basic refractories. Instead, a decrease was found and is attributed partly to somewhat lower temperatures before the oxygen blow is started, but primarily to the reduced slag volume because of the greater refractoriness of the chromite hearth. The increase in the power consumption of HH alloy when melted basic has not yet been explained.

Ferroalloy Grades

The dead melting practice placed strict limitations on the raw materials used in the charge. The inability to remove any carbon from the bath severely limited the carbon content of all alloy purchased scrap to levels which could be diluted down with low carbon virgin materials. Much of the heat resistant scrap alloy available has been used in carburizing furnaces where it very often reaches carbon levels up to 2 pct. A good part of this scrap could not be used in this practice except by very careful dilution.

Table III. Power Consumption in KWH Ton

Practice	HT Alloy	HH Alloy	Overall Production
Dead melt	632	688	693
Acid oxygen	622.5	645	631
Basic oxygen*	569	661	611

* Heats melted on a single shift plant operation. Values have been corrected to compare directly with two shift melting.

This same restriction on carbon drastically limited the ferrochrome and ferromanganese additions to the high priced low carbon grades which materially affected the price required for the finished casting.

The introduction of oxygen in steelmaking was successfully applied to the acid and basic melting of high alloys and resulted in appreciable economies. The relative amounts of the various chromium grades used for all three practices are shown in Fig. 1. It is obvious that oxygen injection has effectively removed the limitation of carbon level. In addition to the substantial saving from using high carbon ferrochrome, further savings in the acid process resulted from replacing a part of the manganese addition with the low cost standard ferromanganese.

Acid melting results in an increase in silicon content, particularly when attempting to super heat the metal near the end of the heat. Further economies were possible provided that the low silicon content in the ferroalloys required as a result of this silicon pickup could be overcome. Basic melting practice permits charging 2.5 to 2.75 pct silicon to finish 1.5 pct. This allowed utilization of the low carbon ferrochrome silicon and the modified 60 to 66 pct Cr high carbon ferrochrome. Very attractive economies resulted because the acid melting limitation of 1.0 to 1.5 pct silicon in the charge had precluded the use of these low cost chrome grades. In addition, the low manganese loss permitted the discontinuance en-

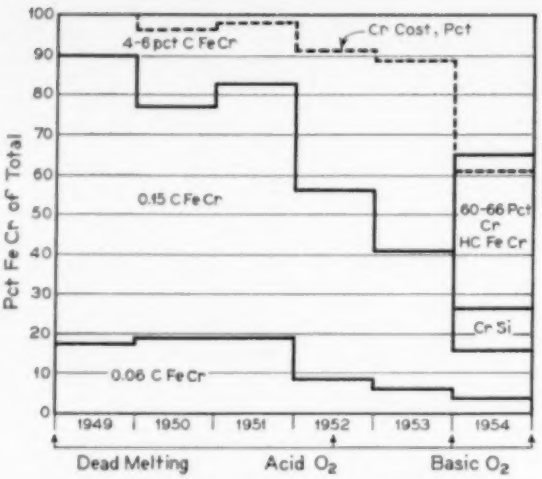

Fig. 1—Relative amounts of the various chromium grades used for three melting practices are compared as well as the percent of chromium cost.

tirely of the low carbon ferromanganese part of the manganese addition to most of the alloy grades.

The degree to which these high carbon ferroalloys can be used depends on practical considerations of refractory life. In this small furnace, where the time at excessive temperature is very short, the practice conceivably could be profitably carried to even greater extremes, by using still higher levels of carbon content in the initial charge and blowing with oxygen for a longer time.

Slag Control

The occurrence of gross slag inclusions trapped on the surface of a casting is highly undesirable in a foundry. Even though the existence of this slag might have no effect on its service life, from a saleability viewpoint it long has been the practice to repair weld such spots if they occur. Under acid melting the frequency with which this trouble arose was quite low and could almost always be traced to using ladles that should have been discarded.

At the end of a typical heat the acid slag was very viscous and voluminous and had a high silica content; the result of the high tapping temperatures required for hand shank-pouring of green sand molds. This caused rather severe cutting of the furnace banks but did result in a slag that was easy to separate from the metal because of its viscosity. Using a modified Herty viscosimeter made with a ⅜-in. diam bore 18 in. long, viscosity tests taken as the heat progressed gave readings of 12 to 16 in. immediately after the bath became flat. This gradually thickened as the bath picked up temperature until a viscosity of 2 to 6 in. was finally reached just before the manganese addition. After manga-

Table IV. Acid Slag Analysis for HH Alloy*

Material	Pct
SiO_2	57
Cr_2O_3	23
MnO	8
CaO	6
MgO	3
FeO-Fe_2O_3	3

* Slag weight equals 20.7 pct of heat weight.

nese the viscosity increased 1 to 2 in. more because of the basic MnO formed.

The coloring of these slags changed from a black green surface with a jet black interior at meltdown to a blue green or emerald green surface with a light blue interior at tap. The standard procedure was to take this slag off in the furnace, tap the heat and add a thin layer of sand to the ladle. This technique gave good, slag-free castings even when pouring without an adequate gating system in the mold. A typical slag analysis for the HH alloy is shown in Table IV.

When the basic process started there was some concern expressed as to slag difficulties on the cope surface of the castings. Viscosity tests were made using the ⅜-in. bore viscosimeter and consistently ran out the full length of the test mold. A ¼-in. bore by 18-in. viscosimeter was made which usually resulted in the slag freezing off at about 15 in. of run, although if the temperature was high enough they could and did run the entire 18 in. This highly fluid slag does enter the castings and is causing much trouble. Present work is aimed at reducing the problem by diligent attention to the ladles and the foundry gating methods. Clean teapot bull ladles rammed with magnesite and new hand shank ladles for each heat, rammed with sand, are helping. But a more basic understanding of the chromite slag is necessary. Peculiarly, the addition of lime and/or other bases does not thicken the slag to any degree. It is believed that the higher lime-silica ratio resulting along with the high silicon level in the bath, merely drives out of the slag some more of the chromic oxide, thus replacing a base with a base and not causing any increase in viscosity.

Table V. Comparison of Alloy Losses for Three Melting Practices

Item	Acid Deal Melt	Acid Oxygen	Basic Oxygen
No. of heats averaged	3215	2853	373
Chrome loss per heat, pct	13.5	14.9	2.1
Overall yield, pct	43.4	47.2	42.2
Overall metallurgical Cr loss, pct	23.2	26.5	4.72
Alloy loss as pct of shipments	7.19	5.01	5.28
Total Cr loss as pct of Cr bought	28.7	30.1	9.59

A very definite advantage is realized in the smaller slag volume. Where the acid process gave a slag weight equal to 20 pct of the heat weight, the basic process has reduced this to 10 pct for the HH alloy and 7 pct for the HT alloy. These slag volumes are in direct proportion to the pounds of chromium contained in the bath.

Reduction of these basic slags is done by silicon in the form of 50 pct lump ferrosilicon or low carbon ferrochrome silicon right after the blow. Further reduction is accomplished by 3 lb of ¼ mesh coke breeze 5 min before tap. The resultant slag is a dull gray black color and is not a disintegrating slag. Considerable alloy is still in the slag, but is unrecoverable when small green sand castings must be poured without pinholes. There is a slight increase in the frequency of gas defects because of the strong reducing period.

Alloy Losses

The loss figures tabulated in Table V show the astounding difference between the practices. The overall economies are in favor of the acid oxygen over the acid dead melt, and the basic oxygen over the acid oxygen by a wide margin. Normally the loss on a per heat basis is the only one shown, but to emphasize the need for conservation of alloys, especially in foundries, the total chrome lost shown as a percent of the total chrome bought is also given, because it shows the compounding effect of yield on this figure.

The equation used for this evaluation was a material balance over an extended time interval. This material balance was made for nickel and chrome separately. Then nickel was used as a tracer to separate the metallurgical loss from the losses in the form of alloy.

The metal loss figures are composites of the entire operation and are considered accurate. In this practice the problem of avoiding contamination from tramp elements has forced the pigging and analysis of all incoming alloy purchased scrap P prior to its use in a heat. Thus the actual pounds of nickel and chrome coming into the melting cycle from all sources is known. The average actual analysis of all shipments S was compiled to obtain the pounds of nickel and chromium leaving the melting cycle. Corrections were made to these figures for changes in inventories of work in process WiP and foundry revert R at the start *1* and end *2* of the test period. The material balance equation expressed as pounds contained chrome where M is miscellaneous alloy sources is as follows:

$$FeCr_{1\text{-}2} + (R_1 - R_2) + P_{1\text{-}2} + M_{1\text{-}2} = S_{1\text{-}2} - (WiP_1 - WiP_2) + \text{Losses}$$

The calculated loss consists of metallurgical loss, unrecovered foundry spills, metal splatter in the slag, slag buttons, and metallics in the grinding dust. After the nickel loss is similarly obtained, it is assumed that no nickel is lost as oxide in the slag, then all nickel losses are in the form of alloy conforming to the average analysis. From this the pounds of nickel and chrome lost in the form of alloy is obtainable. Knowing the pounds of chrome lost as alloy, the difference from the total loss must be the metallurgical loss.

Acknowledgments

The work and advice of H. Peterson, melting superintendent, and the careful interpretation of data by H. Fagan is sincerely appreciated. Permission to publish this paper was given by F. M. Fahrenwald, president, The Fahralloy Co. Appreciation is extended to George Johnson and Frank St. Vincent of the Vanadium Corp. of America for their aid and advice.

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How One Company Appraises Management Development Programs

Do the intangible benefits balance the tangible costs? This company finds the answer definitely is YES

by Carl E. Reistle, Jr.

ENGINEERS as a group are often criticized because they have been responsible for the development of many technical improvements only to allow the administration of them to pass into the hands of others who may have little understanding of how they operate and little competence in their use. These criticisms are not always justified. There are many engineers who have shown themselves to be great administrators in addition to being great creators or builders, and more and more one sees the engineer demonstrating unusual administrative ability as he moves into the executive ranks.

The last five or ten years have seen a new movement developing in this country—a movement brought about through the recognition by business statesmen that successful business administrators and executives just cannot be expected to develop like Topsy, that the management of a company must deliberately take steps to cultivate the proper climate or environment for growth if it wishes to provide an adequate and capable succession of executives in the future. This movement has taken on a number of names—Management Development, Executive Development, Personnel Development—but are these not actually misnomers? Can we develop anybody? In the strictest sense of the word, the author doubts that anyone can change any other person, at least, not much. The change must come from within. Perhaps this movement should be looked at not as a program of developing people but as one of giving people an *opportunity* to develop themselves. In the Humble Oil & Refining Co. there is an Executive Development Program; maybe it might better be called a Program of Development for Executives.

One of the first acts of Humble's original board of directors in 1917 was to set aside capital stock of the newly formed company for purchase by employees, the idea being to make them partners in the business and to encourage career employment. To further this objective the company has also followed a policy of promotion from within, deviating from it in only a few instances. If this policy is to be carried out and if the company is to be successful in its operation, it is obligatory that there be maintained within the company a climate that will stimulate and promote the growth of executive talent.

It has not been easy to do this, as during the 36 years of corporate life there has been a broadening of all industry and a specialization of functions, the like of which the world has never known. The petroleum industry has certainly been no exception to this and, with its many and varied activities requiring highly trained specialists in each of them, presents today a vastly different environment to its executives than it did to their predecessors at the turn of the century. Employees, through no fault of their own, find their work confined to a highly specialized function and when they attain managerial status, are often unable to look on overall problems with the broad perspective required of managers. Thus, it is exceedingly difficult for executives in large corporations to gain through normal experience acquaintance with the many company operations essential to their satisfactory functioning as top-level executives. This has been the experience at Humble, particularly in the case of technical and professional men; and it is also the experience of most engineers.

Recognizing this situation, the company in 1946 set up an Executive Development Program. This was a first attempt to formalize a process that had been going on, after a fashion, in all departments of the company for many years. Broadly speaking, this program consists of appraising abilities of those in executive positions, determining the prospective replacements for such positions, and then providing opportunities for growth and development to those who are considered outstanding. One of the purposes of the program, as stated, is "to establish a uniform system of development and training throughout the company which will assure an adequate supply of potential executives with a broad background of experience." Or putting it another way, the company was setting up a formalized program to make generalists out of specialists.

Within the Humble Co. it is possible to give executives experience in all phases of its operations and to supplement this experience by individual coaching; but a need was felt for something additional which would give these men a broader concept of the administrative function, a recognition of the impact of outside influences on business itself, a better understanding of the role of business in our economic and social systems, and time to reflect upon all of these factors as they are related to our company operations. The company did not feel it was able to do as good a job in these respects as desired, and was aware of the good work that the Harvard School of Business Administration was doing with its Advanced Management Program and so, in the spring of 1947, the comptroller was sent to attend the program and evaluate it for the company. As a result of his enthusiastic participation in the program and upon his favorable recommendation, the company has continued to send executives to every subsequent session. They, too, were of the opinion that the program was of real value and to date 30 Humble executives have attended subsequent sessions. The company would have liked to have sent others in addition to these 30, but because

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been a broadening of horizons and recognition of the rights and aspirations of individuals a better understanding of human emotions and the vital part that they "tolerance" in attempting to describe greater participation in civic and community affairs. And whether or not

of the great demand by other companies for places in the program it has not been possible to do so.

When word came out about similar programs being offered by other universities, they, too, were investigated; and as a result, the company is now participating in the programs of Columbia University, Northwestern University, the University of Pittsburgh, Stanford University, and the University of Houston in addition to the Harvard program. The participants in these programs have given the same enthusiastic reports as given by those who attended the Harvard program, and so today the use of these short intensive courses has become a regular part of the company's Executive Development Program.

Experience shows that those who attend these courses find it difficult to make a clear-cut evaluation of them, especially immediately upon their return to their jobs. They are of the opinion that in a general way the courses have been of value to them; but all of them find it difficult to state specifically, especially to someone else who is unfamiliar with the program, how they have benefited. It has apparently been an experience which they think has been of real value to them, but nevertheless hard to explain. Probably after a year their thoughts and ideas begin to jell sufficiently so that they can begin to explain their feelings in words that convey meaning to others.

The author had the privilege of attending Harvard's Advanced Management Program six years ago. Upon returning to his job he experienced the same difficulty as others in expressing views about the benefits he had derived or could expect to obtain from the stay of 13 weeks there. But now, with the lapse of six years, the benefits the program afforded can be seen rather clearly. Last fall a questionnaire was submitted to the 33 men who attended these various programs prior to 1953. This assured that each would have at least one year of retrospect in which to appraise the program he attended. These ideas concerning the benefits they derived are based on the answers they have given to this questionnaire.

First of all, there has been a broadening of horizons and in some instances an opening up of new horizons. Operations outside of one's own department have attained a new significance and the importance of these operations to the company effort as a whole assumes new proportions. There has been a greater appreciation of the need for statesmanship in business and there is a real desire on the part of people who have attended these courses to measure up to the demands for such statesmanship. Coupled with this has been a more enlightened recognition of the rights and aspirations of individuals working with the company and a real attempt to take these rights and aspirations into consideration when setting up new policies and implementing older ones.

Too often the engineer is prone to look upon the technical and factual side of the problem and when he has worked out his answer, conclude that that is all there is to it. Too often he fails to realize that many of the people in his company without his background, training, and experience do not think as he does, that their attitudes toward the solution of the problem may not be as objective as his, that emotional factors of people must be taken into consideration in addition to the technical or factual aspects of the problem, and that because of his failure to consider these emotional forces quite often many of the people in the organization will not readily go along with the solution. Many technical people who have attended these programs acquired a new method of approach when dealing with administrative problems and they seem to show a better understanding of human emotions and the vital part that they play in any organized activity.

Not only have horizons been broadened but in almost every instance attendance at one of these courses has had a direct and definite effect on work procedures. Nearly all say that they have developed to some extent a more objective approach in the making of decisions, and a willingness to proceed a little more slowly so that the opinions of others and as many facts as possible can be brought to bear upon a problem. Many speak about the need for separating facts from opinions and the patience that is required to do this. Some speak about the overall job of being a manager and the need for anticipating and defining problems before they arise. All of these changes in attitudes should go far in helping these executives develop into capable administrators with real leadership skills.

Emphasis has been placed by many of our country's industrial leaders on the importance of business executives participating more actively in community and civic affairs, but it is unfortunate that so many business men do not understand the need for such participation. Who can say how valuable such participation can be for any company; not so much with respect to the additional business it may gain for itself, as for the generation of a suitable climate in which it can live and carry out its objectives? More than one fourth of the executives who have attended these management development programs say that there has been awakened in them a realization of the need for greater participation in civic and community affairs. And whether or not they recognize this need themselves, it can be observed that nearly all of them, after their return from one of these programs, begin within a short time to show a greater interest in these outside affairs than they had ever shown before. This is of such value to the company that it alone may well be worth the time and money spent.

The good that accrues to industry in general as a result of the impact on the university faculties by the business executives who attend these programs should not be overlooked. Most of the faculty members who work with these executive development programs also teach in the undergraduate and graduate schools of the universities, teach our own sons and daughters, teach the sons and daughters of our employees. Quite often one hears the criticism that university faculties do not have enough contact with business, do not see the practical side of business problems, are too idealistic and live only in a world apart from the actualities faced daily by business men. These criticisms are not necessarily true, but the exchanging of ideas by business men and faculty members in the classrooms of these programs and on the campuses between class periods will do much to bring about a better understanding by faculty members of the problems of business and the effect that Government policies and regulations will have on business. Thus, programs like these play a great part in helping to develop a better climate in which industry in general is able to operate.

The questionnaire submitted to Humble executives asked: "What changes have you noticed in yourself as a result of attending the program?" Three fourths of the answers to this question might be grouped under the heading Greater Tolerance, and it is interesting to see that almost half of these specifically used the words "tolerant" or "tolerance" in attempting to describe changes that they themselves have experienced. Since one of the objectives in the Humble executive development effort is to make generalists out of specialists, it seems that even if a spirit of tolerance is all that a person acquired from one of these programs, the program would have to be considered as worthwhile.

The cost of sending an executive to one of these programs is not low. It represents a real investment in time and money, as he is away from his work for a period of 4 to 13 weeks. Then, too, it is not easy for an executive in the 40 to 55 age bracket to leave home for several weeks and go back to the academic life, nor is it easy for his family to adjust themselves to his absence. In addition to the salary he receives while attending the program, there must be added the cost of tuition, travel back and forth, and living expenses while he is away. All of these add up to several thousand dollars for each individual, and so it is not surprising that the executives hesitate to say positively that they have profited sufficiently from the program to justify the company's expenditure in sending them. But perhaps this is modesty because nearly all of them point out benefits they have received, which should repay the company well for the investments that have been made; and in addition, they are unanimous in stating that the company should continue to send others, provided they are properly chosen. However, there is one definite advantage in sending an executive away for several months that offsets the disadvantages, and that is the opportunity afforded the company to give someone else actual experience in his job, not as an observer but as one who is given the opportunity to carry out the functions of the job and make the decisions, the opportunity to make mistakes and profit from them, and the responsibility of being held accountable for his decisions and actions. Of course, this is an opportunity which can be developed through other means, but it is one that the company has been able to use advantageously.

Now, what are some of the debits in the management development programs? Frankly, with the exception of absence from the job and from home and the matter of cost, all of which have already been mentioned, there are very few. And these few vary in individual cases, depending upon each person's own likes and dislikes. One person might criticize the way the course in Marketing is put on but at the same time speak in laudatory terms about the course in Human Relations, while another will say that he benefited a great deal from the Marketing course but did not care too much for the professor who conducted the discussions in Human Relations. Some say that the course is too long, that the last week could easily be dispensed with, while others say that to shorten the schedule would weaken the course. Naturally, there will be some objections to any of these programs, since it is impossible to suit the individual likes of all who attend them.

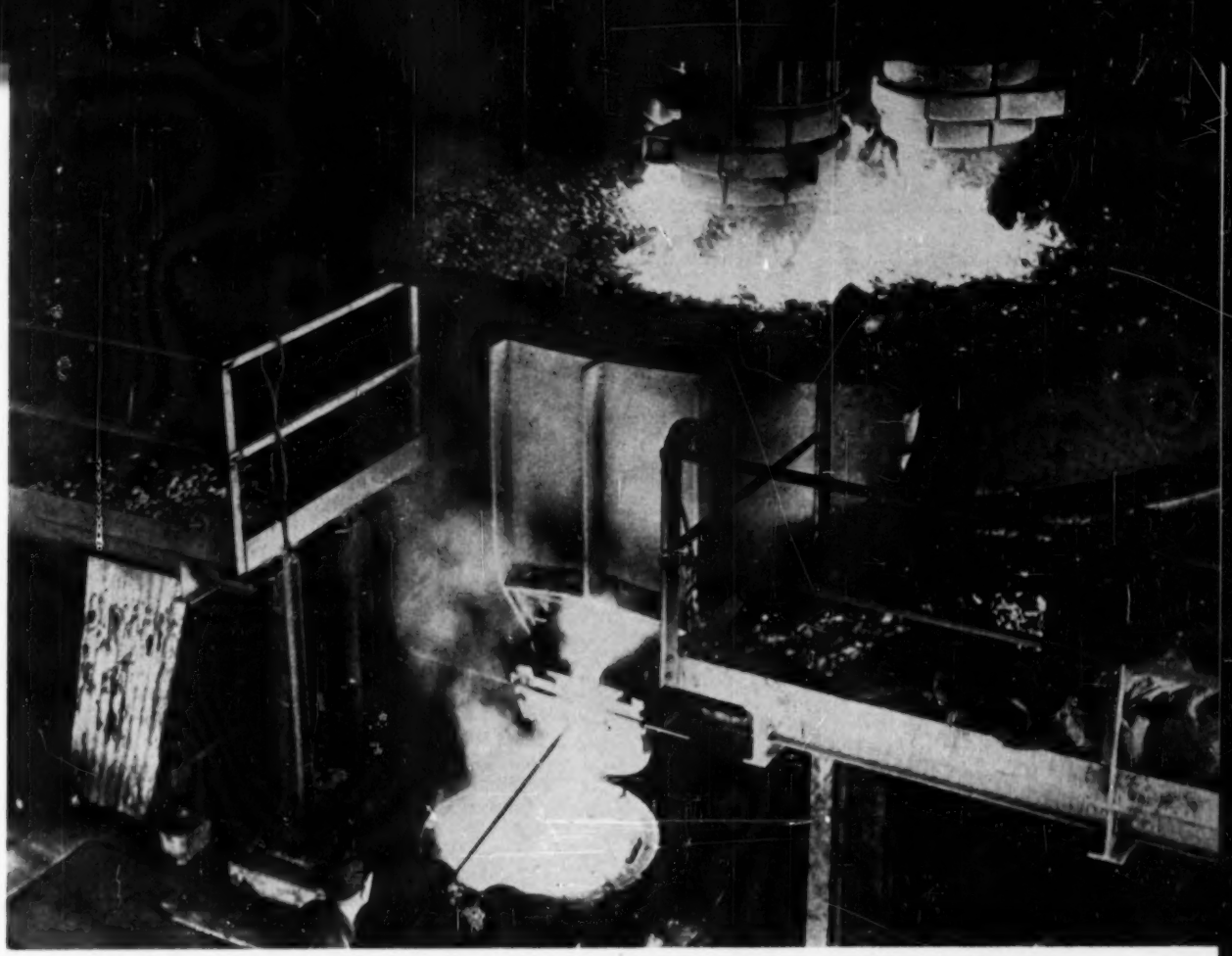
It has been the practice in nearly all of these programs for each class to make frank comments to the faculty, designed to help them improve their program. And it has been by observation that when a preponderance of the participants are unfavorable toward a certain course, methods of instruction, or an instructor himself, steps have been taken to improve the situation. But where criticisms come from only a few and are based mainly on the personal situation, they have not been considered as justification for making changes. This seems to be the correct way of treating comments from the group; and the universities are to be commended for standing fast in meeting their objectives, even in the face of some criticism.

Naturally, it is not feasible to ascertain all of the benefits from these programs that have accrued to each of the executives who has attended, and elaboration of detail on the advantages already mentioned has been avoided. Perhaps the Humble Oil & Refining Co.'s position can best be summed up by stating that it thinks so well of these programs that it is continuing to send executives to them.

However, it should be pointed out that no university has a panacea for all the ills of management nor do all of them follow the same method in approaching the problem of helping management. Each of them is trying to utilize the resources it has available to it and is not necessarily imitating the methods used by others whose resources are different from theirs. And attending each program there will be a group of executives participating just as diligently and just as enthusiastically as other groups in other programs to get all of the possible good out of the program.

Also, it should be suggested that not too much should be expected from those who attend, as all do not respond or react equally. The petroleum industry knows that only one out of every nine wildcat wells drilled finds oil and that the producing wells vary widely in potentiality and production. So it is with executives who attend these programs. Not everyone will get what it is hoped he would, but some will and from these the company will be repaid many times for the investment made.

If the opportunity arises to attend one of these programs, by all means take advantage of it. If the opportunity is not open to you, then find some way on your own initiative to broaden yourself along the lines suggested by these programs so you will be better equipped to take over broader administrative duties and advance in your organization.



Ferrosilicon is tapped from a Pittsburgh Lectromelt submerged arc electric furnace. Electrical resistance and thermal conductivity of electrodes change with increased temperature.

Carbon and Graphite Electrodes Evaluated for Use in Ferroalloy Furnaces

by R. S. Hogue, R. L. Westlake and G. M. Moga

TODAY it is becoming evident that graphite, as well as carbon, must be included in the design and operating considerations in ferroalloy furnaces. The trend to higher powered and bigger furnaces that has caused electric steel melting to adopt graphite is beginning to be felt in the submerged-arc field.

Because the design of the electric furnace should be governed by the conditions desired at the electrode tip, the electrode itself, becomes an important design consideration. The basic principles of electrode selection are the same for graphite and carbon, taking into consideration their fundamental differences in electrical, thermal, and mechanical properties. Table I lists the most important room temperature properties of carbon and graphite. These values show the outstanding differences are in electrical resistance and heat conductivity. Electrical resistance is significantly lower in graphite, permitting higher currents to be carried through the same size electrode. Thermal conductivity of graphite is better and allows a more even temperature to exist throughout the electrode reducing the tendency for spalling and thermal shock.

Two other unique characteristics of the utmost significance are: A—graphite and carbon will not melt, and B—they will maintain their room-temperature strength at least up to 3000°F for carbon and 4000°F for graphite.

R. S. HOGUE, R. L. WESTLAKE, and G. M. MOGA are associated with the Electrode Technical Service Dept., National Carbon Co., Union Carbide & Carbon Corp., New York.

Size of Electrode

In general, an electrode should be of as small diameter as possible and still meet all operating demands. One of the most important of these demands is that the electrode should be large enough to conduct the desired current without becoming too hot.

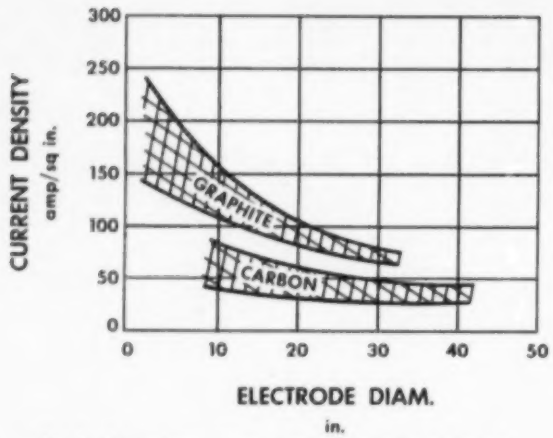


Fig. 1—Approximate range of current carrying capacities of carbon and graphite is shown for various electrode diameters.

As furnaces are designed for more and more power, larger electrodes are indicated, and it might seem that any increase in the cross-sectional area of an electrode would permit a proportionally larger amount of current to be carried. Actually, the practical current carrying capacity, per unit of cross-sectional area, decreases as electrode diameter increases. This is explained by relating the amount of heat generated by the current flow to the speed with which it is dissipated. With the same current flowing, per square inch, a large electrode gets hotter than a small one, because it has proportionately less outside surface to carry away heat by radiation and convection.

The decrease of current density with increased diameter is shown in Fig. 1. The values shown here are only for general guidance. The upper limit of each shaded area may, very properly, be exceeded, if advisable. A lower limit is indicated in line with the general principle of holding the electrode size to a minimum; but, again, other conditions may dictate lower current densities.

The heat generated within the electrode by current flow is supplemented by heat received from the arc tip. This heat is dissipated by radial heat flow to the colder parts of the charge, to the surrounding air, and to the water-cooled electrode holder.

The temperature gradient between the electrode tip and the holder causes axial heat flow to occur. Upward heat flow may be minimized, under certain conditions, by providing a smaller electrode with lesser cross-sectional area, carrying away less heat. In other circumstances, as, for example, where the column extends deep within the charge, a larger electrode seems to be indicated. This is the condition where it is necessary to provide a larger path for heat removal from the electrode tip to prevent overheating at this point. Electrodes in this situation may actually be carrying much less current than they are normally able to carry.

When applying these general rules to help decide the size and kind of electrode to use, a further variable must be introduced. The electrical resistance and thermal conductivity of carbon and graphite are not constant but change with increased electrode temperature.

Fig. 2 shows the thermal conductivity of graphite decreasing with increased temperature. Carbon's heat conductivity increases very slightly over the same range. At a typical operating temperature of about 2800°F, graphite still has twice the thermal conductivity of carbon.

The higher electrical resistance of carbon over graphite is shown in Fig. 3. As temperature gets higher the resistivity of carbon drops and graphite first shows a slight drop followed by a gradual rise.

To try to establish a reasonably low electrode temperature by proper size selection is important because, with all its desirable properties, hot carbon will burn when in contact with air. Even with the most careful selection, the exposed portion below the holder is subjected to air oxidation, which suggests another electrode diameter consideration.

Where the product being made and the operating conditions result in slow-linear electrode consumption, the surface of the carbon is exposed to the air for a longer time than if consumption were rapid. Under these circumstances, a smaller diameter electrode with its faster linear consumption would result in a decreased weight loss from burning, and show less overall electrode use.

Mechanical strength is another important factor to be considered in electrode size selection. A large electrode is stronger than a small one, and the size should be large enough to resist, without breakage, the normal stresses imposed on the electrode columns. Stresses are those caused by movement of the charge as the chemical reaction takes place, and by the control equipment as it raises and lowers the column to maintain proper electrical conditions. This thrust against the side of the electrode, at the lower end, creates a cantilever action in the columns as it is resisted by the clamps and suspension mechanism.

Some final affects of electrode size are felt in the design of certain furnace mechanisms. Smaller electrodes favor lighter, and less complicated, superstructure, electrode clamps, and support mechanism. Less powerful and more sensitive electrode current regulation equipment is possible with smaller electrodes because they are lighter. In the case of covered furnaces, smaller roof ports are possible, providing stronger, and longer lasting roofs. Smaller electrodes tend to favor longer sidewall life by allowing them to operate cooler.

Table I. Typical Room Temperature Properties of Carbon and Graphite Electrodes

Electrode Material	Apparent Density (Gr) Cu Cm	Strength, Psi			Young's Modulus (10^6 Psi)	Specific Resistance, Ohm-In.	Thermal Conductivity Btu per (Hr) (Sq Ft) (°F per Ft)	Mean Coefficient of Thermal Expansion (10^{-7} per °F)
		Tensile	Compressive	Flexural				
Carbon 17 to 45 in. diam	1.57	150	1700	400	0.70	0.0018	8.7	13
Graphite 14 to 35 in. diam	1.53	400	1700	800	0.80	0.00042	70	10

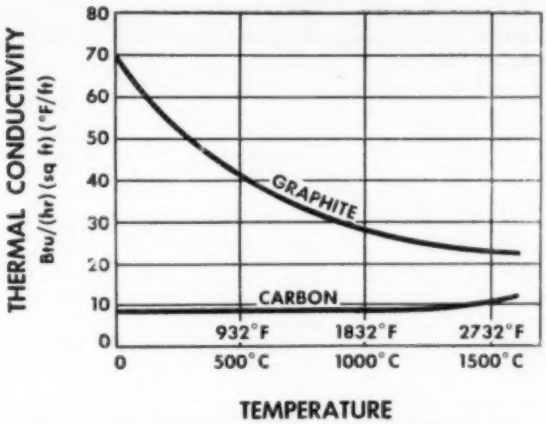


Fig. 2—Effect of temperature on thermal conductivity of carbon and graphite electrodes indicate that at a typical operating temperature of 2800°F graphite has twice the thermal conductivity of carbon.

Of great importance to the chemistry and economics of electric furnace operation is the shape, size and location of the reaction zone. Variables to be considered in establishing this zone are the nature of the reaction, voltage, current, electrical resistance and the depth of charge, particle size of the charge, size and shape of the furnace and arrangement, spacing and size of the electrodes.

Applying the principles of electrode size selection to larger and larger furnaces indicates that graphite may become a popular material in submerged arc furnaces. Very large currents may demand such large carbon electrodes that the higher conductivity of graphite will be used to keep the size within more workable limits from a mechanical point of view. Where a closed furnace is required, the more readily machinable graphite can be held to close diameter tolerance permitting use of tighter seals at the roof ports. Graphite normally will give better electrode consumption than carbon, partly because it requires a higher temperature before it will begin oxidizing: 750°F instead of 650°F. This means that with the use of graphite instead of carbon, there will be fewer electrode additions to the column, less space required for electrode storage and less weight on which to pay freight.

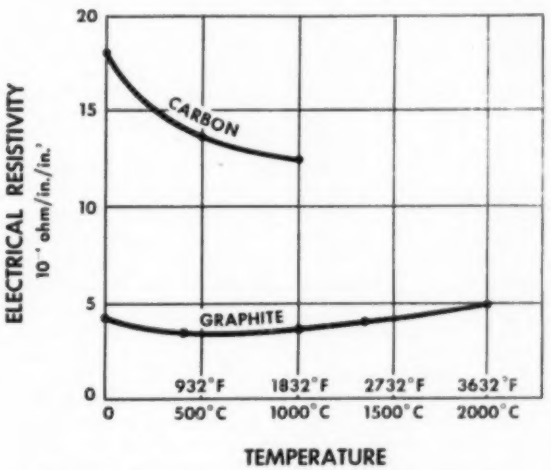


Fig. 3—Temperature effect on electrical resistivity of carbon and graphite electrodes shows that carbon has a higher resistivity.

Of course, the final selection should be made to obtain smallest overall cost considering such items as: A—cost of electrode replacement, B—energy consumption, C—labor, D—first cost of equipment and depreciation, and E—product quality. Fig. 4 shows some electrodes ready for shipment.

Where electrothermal processes require open-arc operation, graphite may be favored in the same way that it is generally favored in open-arc steel melting practice. In operations like this, the best electrode size must be determined occasionally by trial and error since furnace conditions often are unique.

The kind and size of electrodes used for indirect open-arc nonferrous melting has become standardized along with the furnaces themselves. The cylindrical rocking type furnace is essentially the standard in this field and years of satisfactory operation have established certain sizes of graphite electrode as the recommendation for certain capacity furnaces. Electrodes themselves are unique only in that they are made with closer diameter tolerance than normal to permit close fitting electrode ports and to aid clamping of electrodes in the exact center of rotation of the furnace.

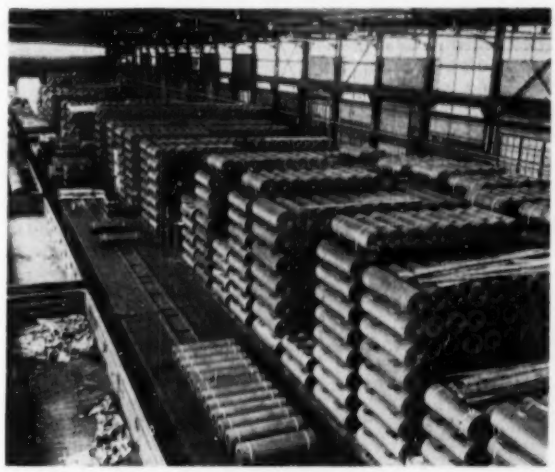


Fig. 4—Electrodes are stored prior to shipment from National Carbon Co.'s plant in Columbia, Tenn.

Electrode Application

To get the best electrode operation requires the application of a few fundamental principles to joint making procedure. The important thing is to produce a joint that will carry current without generating excessive heat or an unbalance of heat between the nipple and socket. A high resistance joint will waste power and the concentrated heat at this joint will increase the rate of oxidation and adversely affect electrode consumption.

A straight threaded nipple is the kind used for joining carbon electrodes. Most electrical and mechanical conditions are satisfactorily met by two standard nipples, graphite or pitch impregnated carbon.

The most severe operations usually demand the graphite nipple which mechanically stronger and will carry more current safely. Less demanding conditions are met by carbon nipples impregnated with pitch which, upon heating, deposits pitch coke in the nipple and at the thread contact. This makes the carbon nipple stronger, decreases the electrical resistance, and prevents joint unscrewing.

Table II. Representative Values for Electrode Consumption, Voltage, Current and Current Density for Various Electric Furnace Products

Product	Electrode Consumption: Approx Lb per Ton of Product	Electrode Consumption: Approx Lb per 1000 Kwh	Voltage Range Phase to Phase	Current Range Secondary Amp	Current Density Amp per Sq In
Ferrosilicon, 50 pct	30 to 40	7	130 to 195	35,000 to 55,000	35 to 55
Ferrosilicon, 65 pct	60 to 90	10	125 to 170	35,000 to 50,000	35 to 60
Ferrosilicon, 75 pct	80 to 130	12	120 to 170	35,000 to 50,000	35 to 60
High Carbon Ferrochrome	30 to 50	10	110 to 200	25,000 to 40,000	25 to 40
Silicomanganese	80 to 100	23	110 to 140	35,000 to 70,000	35 to 70
Ferromanganese	30 to 50	16	85 to 120	30,000 to 60,000	35 to 65
Calcium Carbide	40 to 60	17	120 to 200	30,000 to 80,000	40 to 70
Phosphorus	35 to 70	3	250 to 400	13,000 to 30,000	20 to 30
Refined Copper	4 to 6	19	80 to 190	12,000 to 30,000	80 to 110
Nonferrous Castings	4 to 5	15	100 to 110	800 to 5,000	110 to 300

Where certain applications require the utmost in insurance against joint troubles, graphite nipples are almost invariably required. For certain light duty conditions with short, freely suspended columns, a plain carbon nipple may be used. For plain carbon nipples, it is necessary that a special joint compound be painted on the threads to provide lubrication for the rough carbon-to-carbon contact.

Graphite and pitch impregnated carbon nipples do not require this material but, with them, it is highly recommended that a specially prepared joint cement be used on the end faces only. When properly applied, this cement should provide some lubrication, lower the electrical resistance of the joint, distribute the current more uniformly, increase the mechanical strength, and eliminate joint loosening.

In a few cases, some operators have had success using graphite powder on the end faces. Joints in graphite electrodes generally require no cement because graphite's easier machinability and closer tolerances provide a better fit with smoother threads and end faces.

For a joint to be as stress free and as trouble free as possible, the nipple should be floating in the socket. In other words, the ends should not be tight against either socket base. Sockets are machined deeper than one half the nipple length to allow for this condition, but if the nipple is screwed tightly into one socket, the clearance is effective at only one end. It has been found that the best way to equalize clearance is to put a spacer of some resilient, combustible material, such as rubber, in the socket before screwing in the nipple.

Regardless of the type of nipple or method of assembly, the value of cleanliness of the mating parts cannot be overemphasized. Not only should broken particles from the threads be carefully cleaned away by an air blast, but any accumulated dust should also be thoroughly removed. This dust is such a good electrical insulator that the joint resistance is seriously increased if it is left on the threads and electrode end faces.

None of these methods of joint improvement is fully effective unless adequate torque is applied to tighten the joint. The torque normally recommended is approximately one third that which will break the nipple or socket. If the resistance against tightening torque were plotted, the graph would show the resistance decreasing rapidly at first and then more slowly. The most practical value of torque is at the bend of the curve. On large size carbon electrodes where the recommended torque is high, a mechanical means of joint tightening is ordinarily used. To be successful, these mechanical methods must apply a balanced turning moment requiring that the pressure be applied simultaneously on opposite sides of the electrode column.

To put the proper emphasis on joint making procedure does not mean devoting excessive time to it but, rather, to have a well thought out procedure and all the equipment ready to do a good job. Electrodes should be stored where they are quickly available. Uninterrupted crane service should be available to the top of the furnace. An air line at the storage area will permit blowing out of the top socket before the new electrode is picked up. The heavy nipple must be handled with a clamp or sling and held in a vertical position to allow it to be turned into the socket with ease.

The electrode assembly area must be served with an air line to provide compressed air for the several steps of a proper joint making job. A well designed lift plug will assure ease of handling the new section and will suspend the electrode properly for joining. A chain type pipe wrench with suitably long handle is necessary to turn the electrode. Finally, when necessary to obtain adequate torque, it should be possible to bring the tightening device into action quickly. The use of a feeler gauge is desirable to insure that the electrode interfaces have been brought together tightly. With assembly time reduced to a minimum by following a standardized procedure and by ready availability of equipment, furnace down time will be short and other necessary work can be done at the same time.

By suitable design features, adding electrodes under power is practicable where it is desirable to avoid the short down-time otherwise involved. Likewise, design of the furnace to permit slipping the electrode columns under load is feasible if this feature is important to the overall economics of the operation.

Electrode Performance

A convenient expression of electrode usage is in terms of pounds per ton of product made or pounds per 1000 kwh of electricity used. These figures reflect all of the many variables that can enter into electrode performance so that they are usually expressed as a range of values or as approximate figures.

Table II lists some electric furnace products with representative values for electrode consumption. Also shown are voltages, current, and current density. It is generally agreed that electrode consumption could sometimes be reduced by more specific emphasis on electrode use. At times, this is not always the most economical thing to do and, for instance, in certain cases a greater electrode consumption might be tolerated if tonnage from a specific furnace could be increased.

Sampling Liquid Steel for Oxygen Content:

A Further Evaluation of the Bomb Technique

by S. Gilbert and G. R. Bailey

A further evaluation of the bomb-sampling method for determining the oxygen content of liquid steel is presented. The results of this study and their close agreement with the results of an earlier evaluation confirm the conclusion that the bomb-sampling method is valid for obtaining representative samples for the determination of the oxygen content of a liquid steel bath.

WHAT constitutes the most desirable procedure for obtaining a sample suitable for determining the oxygen content of a liquid steel bath has been a controversial subject for several years. In 1952 Huff, Bailey, and Richards[1] described a modification of the bomb-sampling technique originated by McCutcheon and Rautio.[2] In both the modified and original techniques, a small cast-iron or steel mold containing aluminum wire and appropriately covered is inserted into the furnace, coated with slag, and then plunged into the metal bath. Upon entering the metal, the cover melts and a sample of the steel bath enters the mold.

The data presented by Huff, Bailey, and Richards[1] indicated that reproducible samples could be obtained by their method. Furthermore, the oxygen analyses from samples obtained by their method agreed better with those from controlled laboratory experiments than did analyses from samples obtained by methods previously suggested for sampling large liquid-metal baths.

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Most techniques, other than bomb-sampling, require that a sample of liquid steel be removed from the furnace in a well slagged spoon and poured into an iron mold containing aluminum or into a massive copper mold for rapid chilling. Pouring in this manner requires that the sample pass through the air. The controversy, then, is somewhat as follows:

Proponents of the spoon-sampling method and similar dip-and-pour methods feel that the introduction of a cold steel mold into a molten steel bath may result in a localized boiling condition that may deplete the steel of oxygen to a significant degree in the area where the bomb is introduced. Conceivably this boil could cause results that are, on the average, too low and nonrepresentative of the bath composition. On the other hand, samples that are spooned and then poured through the air may also be nonrepresentative because of oxidation by the atmosphere. The restrictions imposed in sampling with a spoon are much more stringent than those imposed when sampling with the bomb-type mold within the furnace. The spoon sample must be taken in a well slagged spoon, a requirement that depends on the condition of the slag in the furnace. Also, the samples must be poured as quickly as possible, without shaking the spoon and without allowing slag to enter the mold. The necessity of fulfill-

The Transactions papers appearing in the Journal of Metals during 1954 will subsequently appear in AIME Transactions Volume 200, and may be permanently referenced to that volume.

Table I. Sampling Plan: Carbon and Oxygen Content of Duplicate Samples, Pct

	Volume 1, 7.75 Cu In.				Volume 2, 4.00 Cu In.				Volume 3, 3.48 Cu In.				Volume 4, 2.99 Cu In.				Volume 5, 1.80 Cu In.			
	High Carbon		Low Carbon		High Carbon		Low Carbon		High Carbon		Low Carbon		High Carbon		Low Carbon		High Carbon		Low Carbon	
	C	O	C	O	C	O	C	O	C	O	C	O	C	O	C	O	C	O	C	O
Area 1, 6.47 sq in.	0.44	0.0108	0.093	0.0258	0.11	0.0237	0.031	0.0403	0.18	0.0068	0.041	0.0296	0.19	0.0104	0.076	0.0446	0.37	0.0065	0.12	0.0263
	0.41	0.0119	0.085	0.0331	0.12	0.0232	0.038	0.0484	0.18	0.0129	0.039	0.0347	0.20	0.0090	0.057	0.0375	0.31	0.0164	0.14	0.0273
Average	0.42	0.0113	0.089	0.0294	0.12	0.0234	0.034	0.0444	0.18	0.0114	0.040	0.0322	0.20	0.0097	0.066	0.0410	0.29	0.0114	0.13	0.0268
Area 2, 7.85 sq in.	0.14	0.0256	0.033	0.0659	0.18	0.0131	0.11	0.0300	0.33	0.0179	0.13	0.0366	0.10	0.0175	0.041	0.0355	0.35	0.0163	0.048	0.0345
	0.13	0.0238	0.029	0.0701	0.21	0.0126	0.10	0.0376	0.33	0.0208	0.14	0.0265	0.097	0.0187	0.054	0.0379	0.35	0.0204	0.049	0.0392
Average	0.14	0.0247	0.031	0.0680	0.20	0.0128	0.10	0.0288	0.33	0.0194	0.14	0.0266	0.099	0.0181	0.048	0.0367	0.35	0.0184	0.048	0.0368
Area 3, 8.82 sq in.	0.20	0.0158	0.073	0.0320	0.21	0.0150	0.061	0.0310	0.19	0.0101	0.067	0.0428	0.37	0.0141	0.056	0.0432	0.43	0.0081	0.073	0.0349
	0.20	0.0157	0.083	0.0315	0.22	0.0126	0.065	0.0476	0.21	0.0152	0.079	0.0481	0.37	0.0176	0.074	0.0376	0.42	0.0095	0.074	0.0397
Average	0.20	0.0158	0.078	0.0317	0.22	0.0138	0.063	0.0493	0.20	0.0127	0.073	0.0455	0.37	0.0158	0.065	0.0414	0.42	0.0088	0.074	0.0370

ing the foregoing conditions of sampling makes the spoon-sampling method more susceptible to erroneous determinations of bath oxygen concentration than is the relatively simple bomb-sampling technique.

The scope of the present work was to further evaluate the bomb-sampling method by determining whether a carbon boil on the unslagged portion of the bomb has a significant effect on the estimation of the oxygen concentration of the bath. If a carbon boil is produced on the unslagged portion of the bomb-type sampler, it must have a reproducible effect because a high degree of reproducibility between duplicate samples has been consistently obtained with this method of sampling. Since a boil could occur only on the unslagged surface of the bomb and since the magnitude of the boil would be a function of the surface area, then a change in the surface area of the unslagged portion of the bomb should result in a corresponding change in the degree of boiling, and should show a noticeable effect on the estimation of oxygen in the bath. The possibility that the depletion of oxygen in the bath, if it occurs, might be due to a complex relation between the surface area of the mold and the volume of metal sample obtained in the mold must also be acknowledged.

Sampling Program

Accordingly, a series of bomb-type samplers of different geometry was used to obtain 60 samples of open-hearth steel for oxygen analysis by the vacuum-fusion method. Duplicate samples were taken from open-hearth furnaces making high and low carbon steels. The results of analyses for the percentage of oxygen and the percentage of carbon from these samples are shown in Table I. This table lists the individual results of a pair of duplicate samples and their average.

Analytical Procedure

The balanced sampling pattern shown in Table I made it possible to evaluate the effect of the volume and of the unslagged bomb-surface area on the bath oxygen content determined from a bomb-type sample. From the results of the duplicate samples, a further estimate of the reproducibility of the method was derived. However, since all the samples were not obtained at the same carbon content within the particular high and low carbon categories, the data in Table I do not show directly the effects of bomb volume and unslagged bomb-surface area on the determination of the bath oxygen concentration. Any statement made concerning the effect of bomb geometry on bath oxygen concentration also

Table II. Slopes of Lines for Percent Oxygen vs Reciprocal Percent Carbon (C-O Product)

	Volume 1, 7.75 Cu In.	Volume 2, 4.00 Cu In.	Volume 3, 3.48 Cu In.	Volume 4, 2.99 Cu In.	Volume 5, 1.80 Cu In.	Area Averages
Area 1, 6.47 sq in.	0.0020	0.0010	0.0011	0.0031	0.0036	0.0021
Area 2, 7.85 sq in.	0.0017	0.0036	0.0016	0.0015	0.0010	0.0019
Area 3, 8.82 sq in.	0.0020	0.0032	0.0037	0.0020	0.0025	0.0027
Volume averages	0.0019	0.0026	0.0021	0.0022	0.0024	—

Table III. Heights of Lines for Percent Oxygen vs Reciprocal Percent Carbon (Weighted Measure of Oxygen Concentration)

	Volume 1, 7.75 Cu In.	Volume 2, 4.09 Cu In.	Volume 3, 3.48 Cu In.	Volume 4, 2.99 Cu In.	Volume 5, 1.80 Cu In.	Area Averages
Area 1, 6.47 sq in.	0.0300	0.0154	0.0118	0.0270	0.0340	0.0236
Area 2, 7.85 sq in.	0.0257	0.0293	0.0360	0.0180	0.0257	0.0269
Area 3, 8.82 sq in.	0.0284	0.0330	0.0328	0.0326	0.0288	0.0311
Volume averages	0.0280	0.0259	0.0269	0.0259	0.0298	—

will include the accompanying effect of carbon on oxygen concentration. To analyze the data, and to be reasonably certain that the conclusions arrived at are free from any such carbon bias, a method was used which adjusts the oxygen analyses for the percentage of carbon associated with the sample.

A transformation of the data into a linear form, in the range of concentration of carbon and oxygen investigated, was made by relating the percent oxygen to the reciprocal of the carbon concentration. With a linear carbon-oxygen relationship, a method of analysis of the data was used which weights the oxygen analyses properly for the varying percentages of carbon. The data were grouped according to the 15 different mold-geometry combinations possible. This grouping permitted plotting the data as 15 four-point lines. Each line, representing a set of four samples, may be completely described by two parameters: the slope and the height. The slopes of the lines are measures of the C-O product; the heights, adjusted to the average reciprocal carbon content, are weighted measures of the oxygen concentration of the samples included in the calculation of the lines.

Discussion of Results

The C-O products and the weighted oxygen concentrations obtained by relating the percentage of oxygen to the reciprocal of the percentage of carbon are presented in Tables II and III, respectively. Statistical analyses of these data provide the following conclusions:

1—For the bomb samples taken with the bombs of five different volumes, there is no more difference among the C-O products and among the percentages of oxygen than might be expected from the experimental error. Thus, the effect of sampler volume on the C-O product and on the percentage of oxygen of the bath was negligible.

2—For the bomb samples taken with the bombs of three different surface areas, there is no more difference among the C-O products than might be expected from the experimental error in determining the slopes of the lines. The increasing average percentage of oxygen, as measured by the average heights of the lines plotted from the data in Table III, represents an approach to a significant difference among the samples taken with the bombs of the three different surface areas. However, this difference was not sufficient to conclude that bomb-surface area had a significant effect on the percentage of oxygen obtained. In addition, the indicated trend disagreed with the original hypothesis that the oxygen content would decrease as the surface area of the bomb was increased.

3—The slopes plotted from the data in Table II and the heights of the lines plotted from the data in

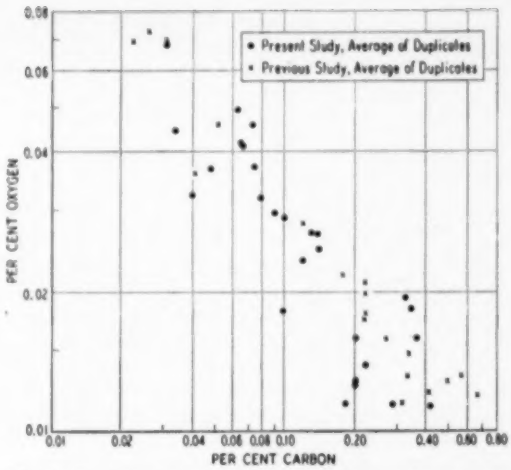

Fig. 1—Percent oxygen vs percent carbon, results of two independent studies.

Table III failed to follow a consistent pattern for increasing surface area over the five levels of bomb volume. This failure, which represents a significant source of variation in the present work, can be attributed almost entirely to the data from the bombs of area 1. The data from the samples obtained with the bombs of area 1-volume 4 and area 1-volume 5 have the greatest variation in duplicate results, which is directly related to the reliability with which their slopes and heights may be estimated. Samples taken with the bombs of the area 1-volume 5 dimensions were the most difficult to obtain in the program. Because of the geometry of these metal samples, they were also the most difficult ones to prepare for carbon and oxygen analyses. Any further attempt to account for the failure of the data to follow a consistent pattern for the volume and area variations in Tables II and III necessarily must include the variations inherent in the open-hearth process itself.

4—The average spread of measured oxygen content between duplicate bombs is 0.002 pct. The average of a pair of such duplicate determinations lies within 0.003 pct of the long-run average at least 95 pct of the time.

In Fig. 1 are plotted the average results of duplicate samples taken with the modified bomb of the previous study[1] and the experimental bombs of the present work. The plot shows that, within the limit of experimental error, the results of the present study agree with the C-O relationships reported previously on the evaluation of the bomb-sampling method.

Summary

The authors believe that the results of this work furnish satisfactory evidence of the negligible effect of a carbon boil, if present, on the estimation of the oxygen concentration of an open-hearth bath by means of the bomb-sampling technique.

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[2] K. C. McCutcheon and L. J. Rautio: Oxygen Samples from Open-Hearth Bath. *Trans.* AIME (1940) **140**, pp. 133-135.

Ionic Nature of Liquid Iron-Silicate Slags

by M. T. Simnad, G. Derge, and I. George

Measurements of current efficiency on iron-silicate slags in iron crucibles showed that conduction is about 10 pct ionic in slags with less than 10 pct silica and about 90 pct ionic in slags with more than 34 pct silica, increasing linearly in the intermediate range. The balance of the conduction is electronic in character. Silicate ions are discharged at the anode with the evolution of gaseous oxygen. Transport experiments show that the ionic current is carried almost entirely by ferrous ions, which may be assigned a transport number of one.

THERE has been increased evidence in recent years that the constitution of liquid-oxide systems (slags) is ionic.[1-3] The principal studies designed to establish the structure of liquid slags have been by electrochemical methods[4, 5] and conductivity measurements[1, 6, 7] which also have indicated the presence of semiconduction in several silicate systems[1, 4-6] and in pure iron oxide.[7] It is well known that many slag-forming metallic oxides have an ionic lattice type in the solid state, and their properties are determined to a large extent by the lattice defects and ion sizes. As Richardson[3] has pointed out, the detailed models of liquid slags cannot be found on thermodynamic data only but "must rest on a proper foundation of compatible structural and thermodynamic knowledge, combined by statistical mechanics."

A careful thermodynamic study of the iron-silicate slags has been carried out by Schuhmann with Ensio[8] and with Michal.[10] They obtained experimental data relating equilibrium CO_2:CO ratios to slag composition and made thermodynamic calculations of the activities of FeO and SiO_2 and of the partial molal heats of solution of FeO and SiO_2 in the slags. It was found that the activity-composition relationships deviate considerably from those to be expected from an ideal binary solution of FeO and SiO_2. However, the partial molal heat of solution of FeO into the slags was estimated to be zero. Their experimental results were correlated with the constitution diagram for FeO-SiO_2 of Bowen and Schairer,[11] with the results of Darken and Gurry[12] on the Fe-O system, and with the work of Darken[13] on the Fe-Si-O system. All these studies were found to be consistent with one another.

The variation of the mechanism of conduction with composition in the liquid iron-oxide-silica system in the range from pure iron oxide to silica saturation (42 pct SiO_2) in iron crucibles was reported in a preliminary note.[14] The current efficiency, or conformance to Faraday's law, showed some ionic conductance at all compositions, the proportion increasing with the concentration of silica. The current-efficiency experiments since have been extended. Furthermore, transport-number measurements have been completed in silica-saturated iron silicates to determine the nature of the conducting ions.

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Experimental

Current Efficiency in Liquid Iron Oxide and Iron Silicates using Iron Anodes: This study was carried out by passing direct current through slags in the range from pure iron oxide to iron oxide saturated with silica (42 pct silica), using pure iron rods as anodes and the iron container as the cathode. A copper coulometer was included in the circuit to indicate the quantity of current passed during electrolysis. Assuming that the cation involved is Fe^{++}, the theoretical quantity of iron lost from the anode according to Faraday's law may be calculated and when compared with the actual loss observed, gives an indication of the extent to which Faraday's law has been obeyed. It also gives an indication of the presence and extent of ionic conduction in the melt.

Preparation of the Slags: About 100 g of chemically pure Fe_2O_3 powder is placed in an iron pot which is heated by induction until the contents liquefy. In this way, FeO is produced according to the reaction $Fe_2O_3 + Fe = 3\ FeO$. More Fe_2O_3 or SiO_2 powder is added and, when a sufficient quantity of molten slag is obtained, the induction unit is turned off, the pot withdrawn, and the molten slag poured on to an iron plate.

Homogenization and Electrolysis of the Slag: **Apparatus**—After considerable development, the setup illustrated in Fig. 1 proved to be quite satisfactory. *A* is an Armco iron cylinder, 1 in. ID and ⅛ in. wall, consisting of three sections placed one on top of the other. The bottom section is a pot about 5 in. long with a small hole drilled in its bottom to allow withdrawal of gases during evacuation of the apparatus. The middle section is 6 in. long and consists of a pot which serves as the slag container, while the top section is a hollow-cylinder continuation of the slag-container pot. The height of this latter section is about 5 in., giving an overall length of approximately 16 in. The iron cylinder is constructed in this way for ease of fabrication, the individual sections becoming welded together after the

first heat. The iron cylinder is centrally located inside a silica tube *B* and supported on a refractory plug *C* which also closes the lower end of tube *B*. The space between the silica tube and the iron cylinder is filled with 10 mesh alumina. The silica tube and its contents are suitably supported by refractory blocks interspaced by a series of radiation shields in a central position in a 48 in. long silica tube *D* which, in turn, is centrally positioned inside a copper induction coil *N*. The lower end of tube *D* is closed by a brass plate *E* sealed into position with apiezon wax.

F is a ¼ in. ID silica guide tube containing a Pt—Pt-10 pct Rh thermocouple suitably protected in an alumina sheath *G*, a weighed pure iron electrode *H*, and its iron suspension wire *I*. The upper end of the guide tube, which is well above the furnace, is sealed with apiezon wax, while the lower end is open, the thermocouple sheath projecting about 1 in. outside, and the electrode about 2 in. The guide tube passes through a number of radiation shields in tube *D* into a pyrex glass head *J* joined on the silica tube by a pair of machined brass flanges and a rubber gasket. This arrangement forms a satisfactorily tight joint. The guide tube finally passes through a ground-glass joint *K* and Wilson seal *L*. In this way, the guide-tube assembly can be moved vertically without danger of the iron electrode touching the side of iron-cylinder *A*. An iron wire *M*, connected to cylinder *A* passing through and sealed with apiezon wax into head *J*, makes the connection to the second pole of the direct-current source. *O* is a sidearm connected by a three-way tap to vacuum and a source of argon.

Procedure—The temperature distribution inside the empty iron pot is first determined. When operating between 1200° and 1400°C, a uniform temperature zone with a variation of less than 10° extends 5 in. above the bottom of the slag container.

About 120 g of the crushed prefused slag is placed in the iron container, calculated for a final slag depth of 2 in., and the apparatus assembled. The system then is evacuated and the induction unit turned on. When the temperature has reached about 1000°C, argon gas is admitted into the furnace through sidearm *O*. The temperature is increased to 1400°C and maintained for a minimum of 4 hr, during which time the slag is stirred with an iron rod at 1 hr intervals. This provides an opportunity for the slag to reach equilibrium with the iron pot and defines the equilibrium $Fe^{+3} + Fe \rightleftharpoons 2Fe^{++}$. At the end of this time, guide-tube *F*, which has been held 1 in. above the slag level during homogenization, is lowered through the Wilson seal until weighed-electrode *H* makes contact with the surface of the slag. This can be observed by a kicking of the ammeter needle in the electrolysis circuit. The electrode is immersed a measured distance below the slag surface and an electrolysis carried out, normally by passing a current of 1 amp for 1 hr. A copper coulometer in the electrolysis circuit indicates the exact quantity of electricity passed during the experiment. At the end of the electrolysis, the electrode is raised out of the molten slag, the induction unit switched off, and the furnace allowed to cool to room temperature under the argon atmosphere. The guide-tube assembly is then lifted out of the furnace at ground-glass-joint *K*, and the electrode removed and weighed.

The change in weight of the electrode is compared with the theoretical change calculated according to Faraday's law, assuming that Fe^{++} ions are taking part in the electrolysis.

The importance of the thermal-gradient effect was observed during experiments with pure FeO in which the zone of uniform temperature was not sufficiently long. The appearance of the iron rod at the end of experiments when no current was applied is shown in Fig. 2. This shows that the electrode dissolved at its lower end and had iron deposited on it at the level of the liquid iron-oxide surface. There is no measureable weight change. The electrode conducts heat away and is at a lower temperature at the level of the FeO surface than at its lower end. When the thermal gradient was removed, the shape of the electrode remained unchanged and there was a negligible loss of weight of the anode with no current applied.

Electrolysis of Silica-Saturated FeO/SiO, Slag using Platinum Anodes: In order to study the anode reaction in greater detail, inert anodes were required. For this purpose, a modified apparatus was built for electrolysis of the slag with platinum anodes.

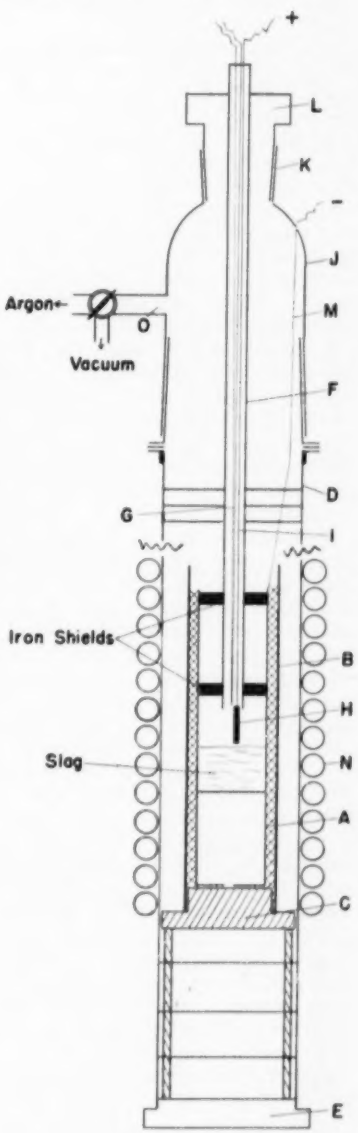

Fig. 1—Apparatus for electrolysis and transport measurements. *A* is Armco iron cylinder; *B*, silica tube; *C*, refractory plug; *D*, silica tube; *E*, brass plate; *F*, silica guide tube; *G*, alumina thermocouple sheath; *H*, iron electrode; *I*, iron suspension wire; *J*, pyrex adopter; *K*, ground-pyrex seal; *L*, Wilson seal; *M*, iron wire; *N*, induction heater coil; and *O*, connection to argon or vacuum.

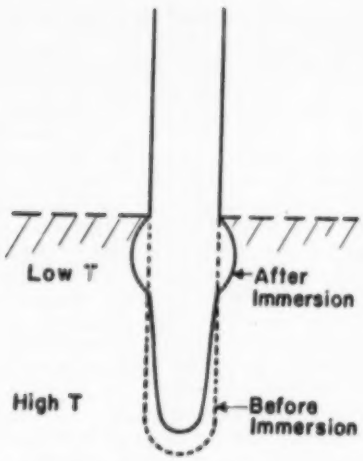

Fig. 2—Change in shape of iron electrode by immersion in liquid iron oxide with thermal gradient: no current applied.

This provided for the collection of the oxygen evolved from the anode during electrolysis. The technique involved the use of a silica capillary tube, 1.5 mm diam, containing a weighed iron lead wire attached to the 2 in. long platinum anode. Any oxygen evolved at the anode oxidized the iron wire, and the gain in weight of the latter could be used as a measure of the quantity of oxygen evolved.

Apparatus—A Kanthal-wound resistance furnace was used for heating the electrolytic cell of Fig. 3. The uniform temperature zone extended over a length of 6 in. The furnace tube, 1¼ in. ID and 48 in. long, was of mullite with ground-glass joints sealed on both the top and bottom of the tube. The bottom joint was connected for evacuating the furnace or filling with argon. The electrolytic cell was lowered through the top openings. The cell was an iron crucible, 2 in. deep and 1 in. OD, containing the slag and held within a molybdenum-sheet tube 1 in. ID and 6 in. long. The latter served to protect the mullite tube from the iron oxide which sometimes formed on the iron crucible. The silica tube containing the platinum-wire electrode and iron lead wire could be lowered or raised through the Wilson seal.

Procedure—With argon flowing through the furnace, the electrolytic cell was introduced into the cold part of the mullite tube. The furnace was evacuated and filled with argon. The cell and silica tube containing the platinum electrode then were lowered into the hot zone. After the slag had melted, the current was turned on and the electrolysis carried out. At the end of the experiment, the cell was pulled up into the cold part of the mullite tube and, after cooling, it was removed from the furnace. The iron wire was removed and reweighed. In all cases, the part of the iron which had been in the hot zone was covered with an oxide layer which tapered off in thickness towards the cold part. It is believed that nearly all the oxygen evolved was converted into iron oxide at low current densities, since no pressure change could be detected in the silica tube during electrolysis. However, at currents above about 2 amp, the evolution of oxygen was too rapid and an increase in pressure was observed. Also, at such high currents, electrical polarization took place as a result of the rapid formation of silica at the anode. This caused the current to fall to extremely low values. At low currents, sufficient iron oxide could diffuse into this layer from the slag to allow the current to flow.

Transport-Number Measurements in Silica-Saturated FeO/SiO$_2$ Slags: This study consisted of the determination of the ions involved and their transport numbers in FeO/SiO$_2$ slags saturated with respect to SiO$_2$ during electrolysis. The method corresponded to the Hittorf technique for measuring transport numbers of ions by measuring the composition change of the electrode compartments after passage of known quantities of current.

Apparatus and Procedure—The apparatus was the same as that described for measuring current efficiencies with iron anodes, see Fig. 1. The only modification in technique consisted of raising the thermocouple and anode inside the inner silica tube 4 in. above the bottom of the tube. The silica tube was lowered into the molten slag, and the pressure of argon in the furnace was then increased until the slag was raised up to the level of the electrode, with about ½ in. of the electrode being immersed in the slag. The temperature at the electrode level was a few degrees higher than in the slag container in order to prevent convection.

A known direct current was passed for a definite period of time at the end of which the silica tube containing the electrode and slag was raised out of the slag in the iron container. After cooling, the part of the tube containing the slag was sectioned into three equal parts, the upper two of which were analyzed separately. In this analysis, the amount of silica in the tube was taken into account by a material balance so that the results would show both a change of slag composition and any solution of silica refractory. Actually, neither occurred as will be shown in the next section. (In trial experiments with platinum anodes, the slag around the electrode was found to change color from the light green characteristic of the ferrous ion to the deep brown of ferric ion in silicates. The level of the color change coincided exactly with the bottom of the platinum wire, and this served to show that convection and

Table I. Cell Efficiency for Electrolysis of Iron Silicate

Composition		Temperature, °C	Current		Anode Loss, Grams	Cell Efficiency, Pct
Pct FeO	Pct SiO_2		Amp	Hr		
100	0	1400	0.00	1.00	0.0050	—
					0.0059	—
					0.0050	—
100	0	1400	2.00	0.50	0.0832	8.3
100	0	1400	2.00	1.00	0.1403	7.1
100	0	1400	1.00	1.00	0.0630	6.3
98.7	1.3	1400	2.00	0.50	0.0985	9.9
98.7	1.3	1400	2.00	1.00	0.1792	9.0
98.7	1.3	1400	1.00	1.00	0.1399	13.9
98.7	1.3	1400	1.00	1.00	0.1228	12.2
94.9	5.1	1400	1.00	1.00	0.1344	13.5
94.9	5.1	1400	2.00	0.50	0.1260	12.6
93.5	6.5	1400	1.00	1.00	0.1103	11.1
93.5	6.5	1400	2.00	0.50	0.1085	11.0
90.2	9.8	1400	0.25	0.50	0.0103	8.3
89.4	10.6	1400	2.00	0.50	0.1401	14.0
89.4	10.6	1400	1.00	1.00	0.0908	9.1
86.2	13.8	1400	2.00	0.50	0.2625	26.3
83.2	16.8	1400	1.00	0.50	0.1638	32.8
82.8	17.2	1400	1.00	0.50	0.2019	40.4
79.3	20.7	1400	1.00	0.50	0.2153	43.1
79.3	20.7	1400	0.50	0.50	0.1058	42.4
77.6	22.4	1400	0.50	0.50	0.1251	50.1
77.6	22.4	1400	1.00	0.50	0.2541	50.9
75.4	24.6	1400	1.00	0.50	0.3019	60.5
75.4	24.6	1400	0.50	0.50	0.1527	61.0
74.0	26.0	1400	0.50	0.50	0.1574	63.1
74.0	26.0	1400	1.00	0.25	0.1666	66.7
65.6	34.4	1400	1.00	0.25	0.2304	92.2
65.6	34.4	1200	1.00	0.25	0.2278	91.1
64.0	36.0	1400	0.25	0.50	0.1112	89.0
64.0	36.0	1400	0.50	0.50	0.2207	88.5
64.0	36.0	1400	1.00	0.25	0.2186	87.5
64.0	36.0	1200	0.50	0.50	0.2303	92.2
57.5	42.5	1400	0.50	0.50	0.2311	92.5
57.5	42.5	1400	1.00	0.25	0.2215	88.6
70.0	30.0	1400	1.00	0.25	0.2002	81.0

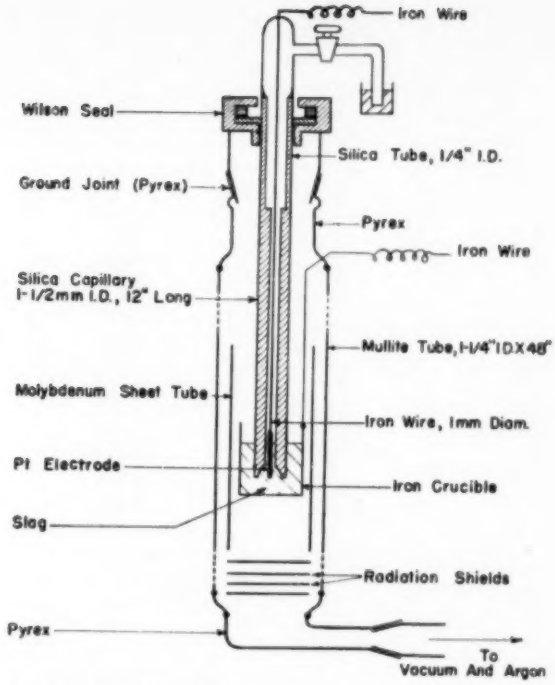

Fig. 3—Schematic section of apparatus for electrolysis of iron silicate with platinum electrodes.

diffusion phenomena in the anode compartment were negligible in this setup).

Results

The essential data for the current-efficiency experiments with iron anodes are listed in Table I. The corresponding variation of cell efficiency with slag composition is shown in Fig. 4. The following conclusions may be drawn from an examination of these results:

1—The current efficiency is between 8 and 12 pct for compositions below 10 pct silica; it increases linearly with silica concentration to about 90 pct efficiency at 34 pct silica; and continues without change to saturation at 42 pct silica. There is, therefore, definite ionic conduction at all the compositions studied, including the pure iron oxide.

2—There is no apparent relation between the composition-current efficiency curve of Fig. 4 and the FeO-SiO_2 equilibrium diagram. However, the break in the curve at 34 pct silica is close enough to fayalite, $(2FeO \cdot SiO_2) = 29.8$, to indicate the existence of some proportion of associated complex ions in the melt above this composition.

3—The current density does not have an appreciable effect upon the current efficiency in the range 0.5 to 4.0 amp per sq cm.

4—The effect of temperature upon current efficiency in the liquid slags is negligible in the range of 1200° to 1400°C.

In the experiments where electrolysis was carried out with platinum anodes, several tests were conducted at higher currents, but the results were not reliable because of polarization and a too rapid rate of oxygen evolution. The results obtained at low currents corroborate those from the series of experiments using iron anodes. In the experiments with platinum anodes, the amounts of oxygen evolved indicate that Faraday's law is obeyed closely. Table II

Table II. Current Efficiency in Silica-Saturated Liquid FeO/SiO_2 with Platinum Anode

Anode	Current, Amp	Time, Min	Temperature, °C	Slag Composition, Pct FeO	Slag Composition, Pct SiO_2	Platinum-Anode Weight Gain, Grams	Weight of Oxygen Evolved, i.e., Iron-Wire Weight Gain, Grams	Current Efficiency, Pct
Pt	0.5	30	1250	57.8	42.2	+0.0001	0.064	89
Pt	0.5	60	1250	57.8	42.0	0.0003	0.120	87
Pt	1.0	30	1250	57.8	42.0	0.0002	0.122	88
Pt	1.0	60	1250	57.8	42.0	0.0001	0.240	87
Pt	1.5	30	1250	57.8	42.0	0.0008	0.180	86
Pt	1.5	60	1250	57.8	42.0	0.0006	0.350	81

shows current efficiencies near 90 pct at low current density. In no case was the platinum anode affected, either by change in weight or in composition. This shows that the anode reaction consists of the evolution of oxygen and the formation of silica as a result of the anodic oxidation of silicate ions, although it does not give information on the complexity of the silicate structure

$$SiO_4^{4-} \rightarrow SiO_2 + O_2 + 4e.$$

The appearance of the slag around the electrode also verified this interpretation. When the platinum electrode was made cathode, a spongy mass of iron was found mixed with the slag near the electrode. The platinum cathode was not affected in this case either, and no oxide was visible on the iron lead wire.

The data obtained from the transport-number measurements are shown in Table III. The anode weight loss from the blank runs, with no current passing, is of the order of 0.1 pct of the weight loss due to current. This establishes the validity of the data for each individual run. Since solution of iron from the anode as Fe^{++} ions approaches Faraday's law and there is no increase in the amount of iron in the anode compartment, an equal amount of iron must have left this compartment by electrolytic migration, i.e., all of the current was carried by this ion. The conclusion is clearly that the transport number of the iron ions, Fe^{++}, is one. This proves that the ionic current is being carried almost completely by these ions.

Discussion of Results

The conclusions drawn from these complete re-

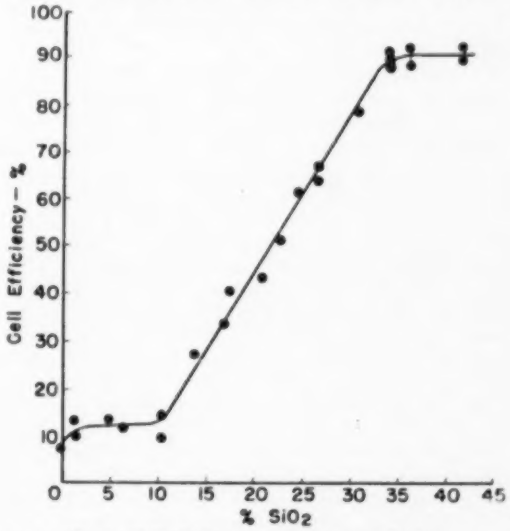

Fig. 4—Cell efficiency in molten-iron silicates.

Table III. Transference-Number Experiments in Liquid Iron-Silicates

Anode	Current, Amp	Time, Min	Temperature, °C	Slag Composition		Slag Composition in Silica Tube				Anode Weight Loss, Grams	Fe⁺⁺ Transport Number
						Top Half		Lower Half			
				Pct FeO	Pct SiO_2	Pct FeO	Pct SiO_2	Pct FeO	Pct SiO_2		
Fe	0	7.5	1300	57.0	42.4	57.1	42.0	57.3	42.1	0.0003	—
Fe	2.0	7.5	1300	57.9	42.3	58.1	41.4	58.4	40.9	0.261	1.0
Fe	0	7.5	1200	57.0	42.3	57.6	41.8	57.5	42.2	0.0002	—
Fe	2.0	7.5	1200	57.5	42.0	57.8	40.9	57.2	42.6	0.259	1.0
Fe	0	10	1300	56.8	42.7	58.8	41.0	57.2	41.9	0.0001	—
Fe	1.5	10	1300	57.6	41.9	58.0	40.9	58.1	41.4	0.2422	1.0
Fe	0	10	1200	55.7	43.5	55.9	43.4	56.0	43.0	0.0004	—
Fe	1.5	10	1200	55.9	43.2	55.5	43.8	56.2	42.9	0.2431	1.0
Fe	0	15	1300	56.8	43.0	57.2	41.9	56.9	43.2	0.0005	—
Fe	1.0	15	1300	57.1	42.0	57.6	42.3	57.0	42.6	0.2538	1.0
Fe	0	15	1200	58.1	41.6	58.3	40.9	57.9	42.0	0.0003	—
Fe	1.0	15	1200	57.2	42.8	58.5	41.3	58.4	41.0	0.2581	1.0
Fe	2.0	7.5	1350	57.4	42.1	57.9	41.8	58.0	41.7	0.2597	1
Fe	0	7.5	1250	57.1	42.1	57.4	41.9	57.3	42.0	0.0005	—
Fe	0	7.5	1300	57.2	42.1	57.4	41.8	57.0	42.4	0.0002	—
Fe	2.0	7.5	1300	58.0	42.1	58.3	40.9	58.2	41.0	0.2578	1
Fe	0	9	1250	55.4	43.7	55.7	43.3	55.0	44.0	0.0005	—
Fe	1.5	9	1250	55.9	43.6	54.8	44.0	55.5	43.5	0.2877	1
Fe	0	10	1280	57.9	41.8	57.6	42.0	57.3	42.2	0.0003	—
Fe	1.5	10	1275	58.0	41.8	58.1	41.6	58.4	40.9	0.2418	1
Fe	0	10	1285	57.0	42.6	57.3	41.8	56.7	43.0	0.0006	—
Fe	1.5	10	1280	56.7	42.8	58.0	41.5	57.2	41.9	0.2427	1
Pt	1.5	10	1250	56.6	42.1	58.1	39.2	56.4	41.0	0.0001	—
Pt	1.5	10	1270	56.6	42.2	60.3	38.9	56.8	41.5	+0.0002	—

sults confirm and amplify those tentatively put forward in the preliminary note:[14]

1—The liquid iron oxide consists of ferrous and ferric ions in an oxygen-ion network, the high disorder of the liquid state providing the equivalent of vacant oxygen sites in the solid crystal, as proposed by Richardson.[15]

2—The break in Fig. 4 at 10 pct silica may be related to the minimum of ferric-ion concentration (2 pct) reached at this composition[11, 12] and to the change in activity of FeO.[9]

3—Above 10 pct silica, the ions resulting from introduction of silica are increasingly effective in the conductance.

4—The break in Fig. 4 at 34 pct silica may indicate the existence of some proportion of associated complex ions in the melts above the fayalite ($2FeO \cdot SiO_2$) composition.

5—The melt consists mainly of ferrous, Fe^{++}, and silicate ions, and also a small proportion of ferric, Fe^{+++}, ions. The ionic current is carried nearly completely by the ferrous ions as shown by transport-number measurements. The electronic current is carried by the mechanism of semiconduction whereby electrons jump from ferrous to ferric ions. No new evidence is given on the nature of the silicate complex.

6—There appears to be a direct relationship between the percentage of ionic conduction and the activity of FeO. The latter has been measured by Schuhmann and Ensio[9] for the same range of slag compositions. This shows that the variation of iron-oxide activity with the weight percentage of silica is very similar to the curve in Fig. 4 showing current efficiency as a function of composition. No variation of A_{FeO} with temperature was established by Schuhmann and Ensio in the range of 1250° to 1400°C, just as the present experiments showed no temperature effect on current efficiency. Schuhmann and Ensio found that the FeO activity-composition relationships deviated considerably from those to be expected for an ideal binary solution of FeO and SiO_2. The partial molal heat of solution of FeO, from the liquid iron-oxide melt in equilibrium with solid iron, into the slags was estimated by them from the activity-temperature data to be zero. As Richardson[16] has pointed out, the entropy of fusion of most slags is not too high so that no major changes in structure are to be expected on melting. Thus the liquid structure resembles the solid, but lacks the long range order. This is in accord with Flood's[17] interpretation that the thermodynamic quantity is necessarily the activity of uncharged molecules or ion pairs (FeO, SiO_2) and that it is rational to connect these quantities with the real constituents of the slags, namely, the ion species.

Acknowledgments

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Inclusions in Steel from Pouring Refractories

by D. J. Carney and E. C. Rudolphy

Large macroscopic nonmetallic inclusions were related to altered fireclay refractories by chemical and microscopic means. Pouring refractories are discussed as a source of these large inclusions. Nozzles and wells are major contributors. Use of alkali and titania contents as tracers is described. Recommendations are offered to minimize the sources of inclusions from pouring refractories.

IN any attempt to improve the cleanliness of plain carbon and alloy steels, consideration must be given not only to the proper deoxidation practice to minimize inclusions, but also to sources of inclusions from pouring refractories. In some cases, the inclusions resulting from deoxidation may be minor in amount, yet a relatively dirty steel may be produced as a result of improper care in the pouring practices for the steel.

The number of studies of inclusions resulting from pouring refractories has increased recently and some excellent reports on this subject have already been published.[1-4] Some of these reports were reviewed briefly in a companion paper by M. P. Fedock.[5] Mr. Fedock's contribution is a fine addition to the studies necessary to improve steel cleanliness for the ultimate benefit of all steel producers and consumers.

In the interest of constantly improving cleanliness, the authors have been actively engaged for the past two years in experimentation to determine more closely the sources of inclusions in steel. One phase of this work was associated with efforts to define more closely the sources of macroscopic inclusions which have been experienced on occasional heats. The term "macroscopic inclusions" is defined in general as those inclusions which are visible to the naked eye or at low magnifications of the order of less than X25. These inclusions sometimes can be seen in the macro-etch tests of billets and slabs. This report summarizes a few of the field experiments and laboratory studies conducted at the South Works of the United States Steel Corp., which were made regarding nozzles, wells, and ingot scums, as well as inclusions occurring in steel products. The data in the present study, together with the data of Mr. Fedock concerning ladle erosion, should be of value in regard to future studies of inclusions arising from pouring refractories. The knowledge gained as a result of data collected thus far at South Works has been helpful in improving the cleanliness of the products and in pointing the way toward further improvements.

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Similarity of Large Inclusions to Refractories and Altered Refractories

In the routine inspection of billet etch tests, occasional inclusions visible to the naked eye have been observed. These large inclusions most frequently occur near the top of the ingot and near the billet surface. When such material was encountered, careful examinations were made in an attempt to determine the origin of these inclusions. In a relatively few cases, examination revealed that the material was unaltered fireclay refractory of a type similar to that used for hot-top material, with only the surface of the inclusions having an altered appearance under the microscope. However, the great majority of these inclusions were completely altered-refractory material. The term "altered refractory" is defined

Table I. Chemical Correspondence of Macroscopic Inclusions with Teeming-Process Erosion Products

Material	Chemical Composition, Wt Pct					
	Al_2O_3	SiO_2	MnO	FeO	TiO_2	Al_2O_3/SiO_2
Nonmetallic inclusion (grade 4620)*	High	Moderate	Moderate	Moderate	Trace	—
Nonmetallic nozzle drip	55.5	24.3	13.1	—	—	2.3
Nonmetallic nozzle drip	48.1	27.5	16.0	—	—	1.7
Nozzle-bore coating (ladle)	39.2	35.5	7.4	—	—	1.1
Nozzle-bore coating (ladle)	36.6	27.5	7.6	—	—	1.3
Nozzle-bore coating (basket)	35.4	44.6	8.1	—	—	0.8
Nozzle-bore coating (basket)	31.5	47.9	8.3	—	—	0.7
Nozzle (unaltered refractory)	30	65	—	—	—	0.4

* Spectrographic analysis: trace, 0 to 1 pct; moderate, 5 to 25 pct; and high, 25 pct or greater.

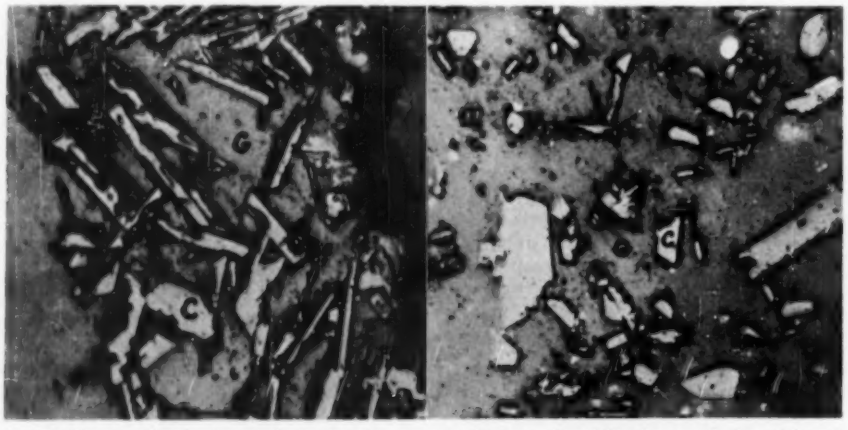
as refractory material, such as fireclay, which has reacted chemically with the liquid steel to such an extent that the physical and chemical characteristics of the refractory have been markedly changed. It has been shown[1-4] that reactions between aluminum dissolved in liquid steel and the pouring refractories (such as $\underline{Al} + SiO_2$ (refractory) = $Al_2O_3 + \underline{Si}$), and between manganese dissolved in liquid steel and the refractory (such as $\underline{Mn} + SiO_2$ (refractory) = $MnO + \underline{Si}$) can, and apparently do, take place along with mechanical erosion. Other similar strong deoxidizers such as titanium and zirconium dissolved in steel can also react chemically with fireclay and silica refractory material. The overall result of the above reaction processes, in the case of aluminum-killed fine-grained steels in contact with fireclay refractories, is to increase the Al_2O_3/SiO_2 ratio of the altered refractory over that of the unaltered refractory. A typical example of this process occurring in practice is given in Table I which qualitatively shows the chemical correspondence of macroscopic inclusions, nozzle drips, and altered-refractory surfaces.

A more clearcut relationship of the inclusion material to that of altered-fireclay refractories was obtained by microscopic examination using petrographic techniques. A typical example of altered-refractory inclusions found in steel and at the surfaces of fireclay refractories in contact with aluminum-killed 0.20 to 0.40 pct C fine-grained steels is shown in Fig. 1. The major constituents of these inclusions are corundum (Al_2O_3), hercynite [(Fe, Mn) $O \cdot Al_2O_3$], and glass. The Al_2O_3/SiO_2 ratio is quite high. As the reaction between the liquid steel and the refractory proceeds, mullite is formed initially. With the addition of more Al_2O_3, corundum is precipitated in two stages. These may be observed microscopically as two different generations. The first occurs approximately isothermally through supersaturation of siliceous glass by the foregoing reaction. The second generation precipitates upon cooling the glass below steel-pouring temperatures.

These studies and observations, made over a period of several years, substantially established the fact that the large inclusions observed in etch tests were not similar to those which might be expected to form during deoxidation of liquid steel but that they were very similar, both in physical and chemical characteristics, to refractory materials which had been altered by reaction with the liquid metal at high temperatures. With this information as a background, the way was pointed for a study of the major origin of this altered-refractory material. This study is described in the sections to follow.

Nozzle Erosion

The chemical and petrographic similarity between the majority of large inclusions and the material which is formed in nozzles during teeming suggested a possible major source of these large inclusions. Further, the nonmetallic drips from nozzles, frequently observed as erosion products during pouring, showed upon examination that corundum, spinel, and glass are being carried into the ingot. Chemical



a—Altered-fireclay nozzle-bore surface.

b—Macroscopic inclusion in 8620 steel.

Fig. 1—Micrographs showing similarity in microstructure between altered-fireclay nozzle bore (*a*) and macroscopic inclusions (*b*) in 8620 steel. *C* is corundum; *G*, glass; and *H*, hercynite-rich spinel. Unetched. X500.

Table II. Chemical Correspondence of Ingot Scums and Nozzle Drips Formed During Teeming of Plain Carbon, Fine-Grained Steels (0.20 to 0.50 C; 0.70 to 0.90 Mn; 1¼ Lb Al per Ton)

	Chemical Composition, Wt Pct		
	Al_2O_3	SiO_2	MnO
Nozzle drip	42.9	24.7	12.0
Nozzle drip	53.7	30.4	5.2
Nozzle drip	41.3	27.8	6.6
Ingot scum	45.2	22.8	15.1
Ingot scum	32.0	20.5	23.5
Ingot scum	46.4	29.3	11.6

compositions of some nozzle drips and ingot scums are given in Table II. The similarity is striking.

Inspection of standard fireclay nozzles before and after use revealed that marked erosion of nozzles occurs during teeming. One example is shown in Fig. 2. The majority of alloy steels produced in the open hearth at South Works are poured from a standard pit ladle into a so-called "basket" which is equipped with two sets of nozzles and stopper-rod assemblies and which has a capacity of approximately 20 tons of metal. The nozzles examined were basket nozzles. The enlargement of the nozzle bores from 2 to 2½ in. during the teeming operation furnishes approximately 15 cu in. of refractory material which is eroded throughout the whole heat and carried into the molds. Many nozzles examined eroded more than this amount during teeming.

As a result of the above examinations, it was decided in 1948 to conduct a trial with nozzles made of a highly refractory aluminous material in an effort to minimize chemical and mechanical erosion. In the trial, one basket nozzle was mullite and the other was of standard fireclay composition. A fine-grained 8620 heat was teemed through this basket. After teeming, both nozzles were carefully removed for detailed examination. The standard nozzle was eroded in a manner similar to that shown previously in Fig. 2. The mullite nozzle showed no erosion. However, close examination revealed that a large amount of nonmetallic material was carried down into the mullite nozzle. This is shown in Fig. 3. These facts strongly suggested that the nozzle well and the ladle brick were also sources of altered-refractory material. With the mullite nozzle, there was an absence of fluxing of the surface which prevented erosion and subsequent entrainment of the refractory material in the molten steel. In fact, metal chills and eroded nonmetallic material from above the nozzles remained as an accretion on the refractory surface of the mullite nozzle. While the highly refractory nozzle markedly reduced the amount of nonmetallic material entering the mold, it also constricted the metal-stream flow and produced a ragged-spraying stream. On a practical basis, the use of mullite nozzles is limited by these factors. A compromise suggestion for nozzle material based on these trial results is suggested later in the paper.

Fig. 2—Erosion of standard fireclay nozzle during the pouring of a fine-grained heat. The white lines designate the original bore contours of the nozzle. Area reduced approximately 45 pct from ⅞ actual size for reproduction.

The foregoing discussion has been centered around the nozzles, since they definitely are subject to the most severe erosive conditions as the molten steel moves rapidly over their surfaces. However, since it was evident from these tests that the well material was also a fruitful source of refractory inclusions, a further study of this factor was indicated.

Well Erosion

The clay-shale which is used as standard practice in the basket-ladle wells contains a high percentage

Table III. Chemical Composition* of Nonmetallic Materials Obtained from Ingots of 8620 Steel Poured During High Refractory Well Trial

	Na_2O	K_2O	Al_2O_3	SiO_2	TiO_2	FeO	MnO
Clay-shale	Low	Moderate	High	High	Low	Moderate	Trace
Ingot scum (sillimanite well)	Trace	Trace	High	Moderate	Trace	Moderate	Moderate
Ingot scum (clay-shale well)	Low	Moderate	High	Moderate	Trace	Moderate	Moderate
Material from top discard (sillimanite well)	Trace	Trace	High	Moderate	Trace	Moderate	Moderate
Material from top discard (clay-shale well)	Low	Moderate	High	High	Low	Moderate	Moderate

* Spectrographic analyses: trace, 0 to 1 pct; low, 1 to 5 pct; moderate, 5 to 25 pct; and high, 25 pct or greater.

of free silica and has a fusion point low enough to permit it to be eroded readily. In order to check the effects of erosion on this material, one well of a basket was made with a refractory sillimanite facing. The other well was made with the standard clay-shale practice. In order to eliminate the nonmetallics originating from the nozzle, mullite nozzles were used in this basket. Top discards were obtained from one ingot poured through each nozzle of the basket from each of two heats of 8620 steel. In these two heats, no large nonmetallic material was observed in the discards from the ingots which had been poured through the high refractory (sillimanite) well. Nonmetallic material high in Al_2O_3 was observed in the discards from the ingots which had been poured through the clay-shale well.

The clay-shale used in these wells contains considerably more alkali than any of the other materials encountered in the teeming practice. As such, the alkalis formed good tracers for studying well erosion. The ingot scum and material from top discards of ingots poured through the sillimanite well contained only traces of alkali, whereas corresponding materials from ingots poured through the clay-shale well contained notably higher alkali contents (see Table III). It must be concluded that the well, in addition to the nozzle, contributes nonmetallics to the ingot.

The extent of erosion by the liquid steel flowing above the well was not known other than that some nonmetallic material is carried across the well and nozzle surfaces. Evidence of this can be seen when the latter are made of sillimanite and mullite. Away from the well, it is expected that the erosion of brick materials quickly becomes less significant because the velocity of steel flow decreases markedly. "Dirt," or partially adhering patching cements can be carried along with the steel into the ingot. Mr. Fedock's recent work established that the erosion above the well which contributes to refractories in the ingot is limited most probably to an area centered on the well and no greater than 20 in. in diameter.

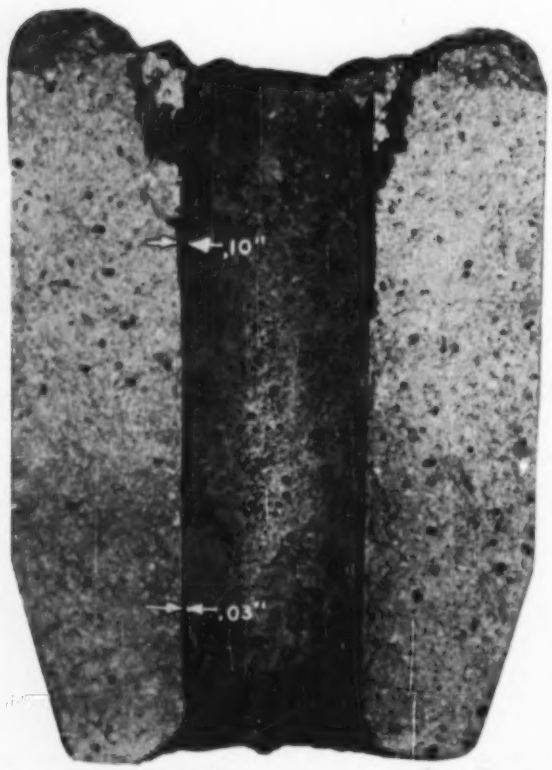


Fig. 3—Mullite nozzle showing an accretion of nonmetallic material which has been carried into the nozzle bore from the refractory-steel surface above. This material would have been carried into the ingot had the nozzle been subject to normal erosion. Area reduced approximately 30 pct from 4/5 actual size for reproduction.

Table IV. Use of TiO_2 Content as a Tracer of Refractory Erosion and Alteration in Liquid Steel

			Chemical Composition, Wt Pct				
		Grade	Al_2O_3	SiO_2	MnO	TiO_2	Alkali
Ingot scum	Altered refractory	8620	44.2	29.8	13.0	1.04	1.4
Ingot scum	Altered refractory	8620	45.4	24.4	14.0	0.80	1.1
Ingot scum	Altered refractory	8620	42.7	27.3	18.0	0.76	0.9
Nozzle drip	Altered refractory	8620	55.5	24.3	13.0	1.07	—
Nozzle drip	Altered refractory	8620	42.9	24.7	12.0	0.96	—
Nozzle	Unaltered refractory		28.7	61.8	—	1.6	3.4
	Unaltered refractory		34.0	54.4	2.3	1.88	3.2

Use of TiO_2 as a Refractory Tracer

Altered and unaltered refractories have been related to the large macroscopic inclusions in steel, and origins for these inclusions have been suggested. The extent to which refractories were responsible for these corundum-glass inclusions has yet to be considered.

Almost all refractories contain 1 to 2 pct TiO_2. Furthermore, the titanium content of aluminum added to the steel as a deoxidizer is almost nil (0.07 pct). Other addition agents also are very low in titanium. The presence of TiO_2 in nonmetallic inclusions in steel, therefore, would indicate that the material had a refractory origin. The amounts of TiO_2 in several samples of nonmetallic material associated with the teeming process, namely ingot scums and nozzle drips, are presented in Table IV. The chemical composition and microstructure of these samples were similar to corundum-glass inclusions in steel (Fig. 4). Since more than 0.75 pct titania was present in these materials, it must be concluded that they contain some refractory. The amount of refractory present in these cases was 50 to 75 pct, assuming no gain or loss of the TiO_2 present. It was not determined how much of the remaining altered material was a reaction product between refractories and steel and how much was a deoxidation product from the steel.

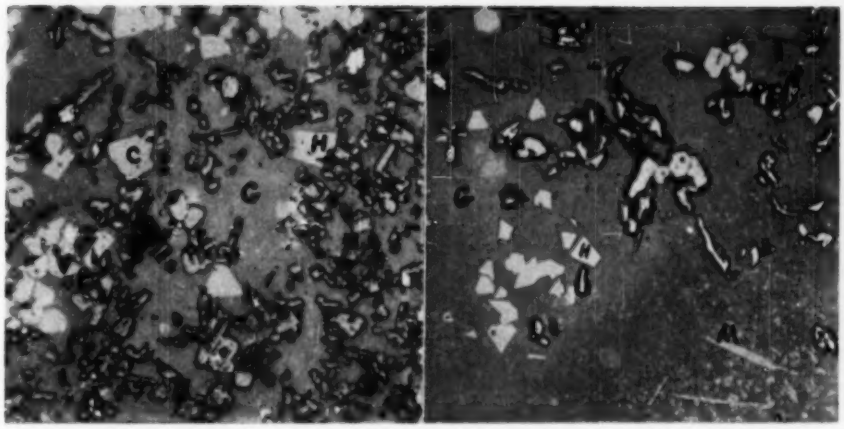
Summary and Recommendations

A brief summary of the major findings of this report is as follows:

Chemical and mechanical erosion of fireclay and silica pouring refractories by killed steels results in altered-refractory material being carried into the ingot. This material occasionally is entrapped in the ingot body and results in large macroscopic inclusions which are sometimes visible to the naked eye in billet etch tests.

A major source of these inclusions was found to be the nozzle. It is estimated that the standard fireclay nozzle contributes as much as 15 cu in. of refractory material to the ingots during the teeming

a—Ingot scum.



b—Macroscopic inclusion in 4620 steel.

Fig. 4—Micrographs showing similarity in microstructure between ingot scum (*a*) and macroscopic inclusion (*b*) in 4620 steel. *C* is corundum; *G*, glass, *H*, hercynite-rich spinel, and *M*, mullite. Unetched. X500.

of a single heat. A trial conducted with more refractory mullite nozzles reduced this erosion to a minimum but contributed to a flared pouring stream.

The use of a clay-shale-well practice also contributes a large amount of refractory material to the ingot. The use of a sillimanite-well practice rather than the clay-shale practice greatly minimized refractory erosion in the well area on low alloy steel heats. Tests revealed that the number of inclusions observed in ingot-top discards were considerably less for the sillimanite-well practice than they were for the standard clay-shale-well practice.

The use of TiO_2 as a tracer material further confirmed the fact that portions of ingot scums, nozzle drips, and certain inclusions definitely had refractory origins.

It might be logical to inquire at this point, in light of preceding information, how clean steels are ever produced, or rather, why more difficulty has not been experienced in the past with inclusions from these sources? The answer to these questions may be as follows:

Once the nonmetallic material is carried into the ingot, it has a tendency to rise out of the molten steel because of its lower specific gravity. The accumulation of ingot scum on the top of the ingot is evidence of this density separation. Of ten corundum-bearing inclusions observed in etch tests, six were in the top cuts. Nonmetallic material from ingot scum is frequently observed as partially altered refractory on the surface of stripped ingots. These were certainly entrapped in the chill zone at the molten-steel surfaces during teeming. Material in this zone normally is removed from the ingot with the scale before rolling. Material further in from the surface must be entrapped in the solidification front. To date, the most plausible explanation is that the teeming stream carries scum below the surface of the ingot after the chill zone is formed (particularly during topping of the ingot). The shoulder at the base of the hot top possibly becomes a lodging place for some of that material. Thus, the mechanism of entrapment of inclusions below the surface of the ingot must be investigated thoroughly before complete elimination of large inclusions can be attained.

The following recommendations are offered to minimize sources of inclusions from pouring refractories:

Mold Cleanliness: The importance of mold cleanliness is well known. All refractory material should be removed. It cannot be overemphasized because any nonmetallic in the mold automatically becomes a potential source of entrapped inclusions.

Basket Cleanliness: Material is carried into the well from the basket floor. Partially adhering slag, slurry, or patching materials on the bottoms would be especially subject to erosion by the molten steel.

Nozzles: Snow and Shea[4] have shown that high heat duty fireclay nozzles are less subject to erosion by fine-grained low manganese steels than are conventional nozzles. Such nozzles may have merit. A trial of these nozzles is recommended along with a change in nozzle-bore shape. Considerable refractory is eroded by the stream because the bore shape does not conform to the stream flow pattern. While the bore of a mullite nozzle likewise could be changed to give a less ragged stream, the use of mullite nozzles currently is not recommended because of their cost and their tendency to build up an accretion in their bores.

Wells: The adoption of the sillimanite-well practice for baskets is recommended over the clay-well practice for steels where macrocleanliness is important. A preformed fired well block in the basket is suggested as an alternate procedure. Its installation and maintenance may be easier than the sillimanite well.

Acknowledgment

The authors would like to acknowledge the work of L. H. Van Vlack and R. G. Wells, former members of the South Works Research Laboratory Staff, who collected the data for this report.

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Analysis of Factors that Limit the Production Rate And Coke Rate in the Iron Blast Furnace

by W. O. Philbrook

An engineering analysis indicates that the coke rate in present blast-furnace practice is set not by chemical or thermal needs but to give adequate charge permeability for economical driving rates. An equation showing interrelations among pressure drop, gas flow, and charge characteristics has been derived. Flooding conditions and the location of the fusion zone are discussed. Maximum ore size is probably limited by chemical reducibility rather than by heat transfer.

THE intention of this paper is to show that the limitations on the production rate and coke requirement of the iron blast furnace are not primarily chemical or thermal in nature but are related to characteristics of the burden and operation that govern gas flow, heat transfer, and stock movement. A semiquantitative analysis will be developed in terms of standard engineering principles to give a useful picture of the relative importance of physical and mechanical factors in contrast with the chemical features of the process that are more commonly emphasized.

Most of the details of calculation have been omitted in the interest of brevity and to avoid submerging the main points in a sea of arithmetic. Numerical values have been arrived at by objective methods from reported operating data or by reasonable estimates of material properties where exact data are lacking and then tested to see how well they fit practice, rather than by working backward to find what assumptions are needed to give the desired answers. The analysis is oversimplified in many ways because exact engineering relations have not yet been studied for bed distributions as complex as are known to exist in the blast-furnace stock column. One objective of this paper is to show that the blast furnace, in spite of its complexity, can be attacked by engineering methods in the hope that this viewpoint will stimulate the kind of research needed for a more exact treatment.

The basic ideas and many of the conclusions presented are qualitatively well known. A careful literature survey to credit the source of all observations not original to the author would indeed be voluminous, and such has not been attempted. T. L. Joseph's Howe Memorial Lecture[1] gives an excellent background of what is known about the internal workings of a blast furnace. Other important reference works or sources of specific information will be noted. It is hoped that the present paper will contribute something new, in method of attack and interpretation, that will aid in a clearer understanding of how to operate the blast furnace to best advantage.

Factors Limiting Production and Coke Rates

The rate of combustion of coke in a blast furnace, as in any fuel bed, is exactly proportional to the rate at which air is supplied. It is axiomatic, then, that the production rate of a furnace is limited by the rate at which it will take wind, i.e., burn coke, and the amount of coke that must be burned at the tuyeres to make a ton of iron of suitable composition. These two possible limiting factors will be considered briefly in the order mentioned.

One common restriction on wind rate is flue-dust production. Since the volume of top gas is nearly proportional to the blast volume (by a factor of about 1.35 on a dry basis), an increased wind rate causes a correspondingly increased linear velocity of top gas in the throat of the furnace. The higher

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rising velocity increases flue-dust losses until a halt must be called for either economic or mechanical reasons. Even for coarse burdens where flue dust is not a serious problem, however, too high a flow rate through the stock column ultimately leads to instability of one sort or another so that the furnace will no longer move smoothly. This sort of limitation on wind rate is one of the main variables that will enter into the later discussion.

The minimum coke rate required to make a ton of iron seems to be a more complicated problem. Certain fairly obvious factors may be dismissed as secondary variables that need not complicate the present discussion. For a basic iron, about 125 lb of carbon is required in the hearth to carburize and desulphurize the iron and to reduce silicon, manganese, and phosphorus. High gangue content of the ore and high ash and sulphur in the coke obviously will require more slag and flux, which entails higher coke consumption for heat and probably higher solution loss because of the CO_2 from the extra stone. Since radiation, convection, and cooling-water losses are relatively constant with time, the cooling losses per ton will vary inversely with the production rate in tons per hour. The foregoing factors will be ignored in order to concentrate attention on the role of carbon in the main process of reducing ore.

Functions of Coke in the Blast Furnace: By a process of elimination, an attempt may be made to select the factor limiting the coke rate from among the three recognized functions of coke in the blast furnace: 1—Coke supplies the chemical reducing agent for smelting ore. 2—Coke is the source of heat for the process. 3—Coke, as a large-sized material occupying 60 pct or more of the volume of the stock column, provides the necessary permeability for gas flow. The common statement that "coke supports the burden" is merely a recognition that the coke must be strong enough to retain its size during its passage down the shaft. In principle, any of these functions might control the coke requirement. The third one has received little attention as a limiting factor, while controversy has raged over the other two. A comparison by pairs will be made to see which can be eliminated from consideration.

Fuel vs Reducing Agent: Grüner in 1870 proposed (see discussion in ref. 2) that in an "ideally perfect working" blast furnace all reduction should be done by carbon monoxide, or by what is known as "indirect" reduction. Although Grüner's theorem has been thoroughly discredited,[3] it died slowly and was a substantial roadblock to correct thinking about blast-furnace operation because of its superficial plausibility.

One of the difficulties involved in estimating the amount of carbon required for reduction has been a decision on the limiting CO/CO_2 ratio set by equilibrium considerations. Austin[3] calculated the theoretical CO required for *isothermal* indirect reduction of Fe_2O_3 at various temperature levels and found that the process might be carried out with CO/CO_2 ratios little higher than 1 if the reduction could be completed at a relatively low temperature. He made no allowance, however, for carbon consumed in hearth reactions or for the presence in the furnace of coke and of CO_2 from limestone.

Gumz[4] has proposed that the rising gases in a blast furnace actually reach heterogeneous equilibrium with the solid phases in the shaft at some "reaction temperature." This temperature is determined by a simultaneous solution for the fundamental gas-solid reaction equilibria and the heat balance up to the top of the "reaction zone" at which the reaction temperature occurs. He calculated a reaction temperature of 675°C (1247°F) for a charcoal blast furnace and assumed that this temperature was typical of other (coke) furnaces working at ordinary pressures. Equilibrium was established on coke as the bulk phase rather than on iron oxides. The CO/CO_2 ratio at the reaction temperature was 2.32, but this was lowered somewhat in the preheating zone above the reaction zone by first-stage reduction of Fe_2O_3 to Fe_3O_4, calcination of carbonates other than $CaCO_3$, and drying of the charge. Gumz's procedure allows for no influence of nonuniform stock distribution or of the physical structure or properties of the charge other than composition. In view of the known segregation of charge materials within any present-day blast furnace, the top-gas analysis can be nothing better than an average of several poorly mixed streams rising at different temperatures and velocities through zones of widely varying ore and coke content. It seems pointless to belabor equilibrium considerations.

For practical purposes, a top-gas CO/CO_2 ratio of 2.0 can be taken as representative of average practice and 1.5 as the lower limit likely to be attainable in real practice. Allowing for the carbon consumed in hearth reactions and for enough $CaCO_3$ to flux silica from the ore and coke, the coke requirement, shown in the upper part of Table I for purely "indirect" reduction, is reached. Since many American furnaces are operating with lower coke rates than calculated, it is obvious that the coke needed for indirect reduction according to Grüner's theorem cannot be a limiting factor, an observation that has been pointed out by numerous writers.

Since some "direct" reduction or "solution loss" is recognized as beneficial in reducing the coke requirement, the argument now may be shifted to the opposite extreme. Consider a smelting furnace in which heat can be supplied independently of the coke, as by electrical energy. A practical equivalent is the electric-smelting furnace used in Italy and the Scandinavian countries or a ferroalloy furnace. Under these circumstances, there is no need to burn coke at the tuyeres to supply heat, and it can be used most efficiently as a reducing agent. The mechanism of reduction will undoubtedly be a gas-solid reaction, that is, indirect reduction; but the continuous regeneration of CO from CO_2 by carbon will give the same net result as though direct reduction had

Table I. Coke Required for Indirect and Direct Reduction at Various Top-Gas Ratios, per Ton of Iron

Top Gas, CO/CO_2	Net Reaction with Ore	Lb $CaCO_3$	Lb Coke
Indirect Reduction			
1.5	$Fe_2O_3 + 7.5\ CO = 2\ Fe + 4.50\ CO + 3.00\ CO_2$	850	1940
2.0	$Fe_2O_3 + 9.0\ CO = 2\ Fe + 6.00\ CO + 3.00\ CO_2$	900	2300
Direct Reduction			
2.0	$Fe_2O_3 + 2.25\ C = 2\ Fe + 1.50\ CO + 0.75\ CO_2$	700	710
2.5	$Fe_2O_3 + 2.33\ C = 2\ Fe + 1.67\ CO + 0.67\ CO_2$	700	730
3.0	$Fe_2O_3 + 2.40\ C = 2\ Fe + 1.80\ CO + 0.60\ CO_2$	700	760

Assumptions

Ore: 52 pct Fe, 8 pct SiO_2, natural basis.
Coke: 85 pct FC, 5 pct SiO_2, natural basis.
Iron: 4.5 pct C, 1.0 pct Si, 1.75 pct Mn, 0.25 pct P, 92.5 pct Fe. 120 lb C required for hearth reactions, neglecting desulphurization.
Flux: Enough $CaCO_3$ supplied to give pct CaO/ pct SiO_2 = 1.4 in slag. CO_2 from flux reacts with carbon by solution loss to adjust to stated CO/CO_2 ratio, including CO from hearth reactions.
Results rounded to nearest 10 lb.

taken place. It is convenient to write the reaction as though solid carbon were the reducing agent. Recognizing that such a process might not utilize CO as efficiently in the upper part of the shaft as the conventional furnace, top-gas ratios of 2.0 or higher may be assumed. Again taking into account the hearth reaction and flux requirements, the coke rates shown in the lower part of Table I may be calculated. It seems evident that, if it were not for the necessity of generating heat at the tuyeres, the coke could be cut well below present levels by a greater amount of "solution loss." Clearly, the reducing-agent requirement is not the factor that limits the coke rate.

It must be emphasized that the foregoing argument has assumed that hearth temperature is controlled to produce a specified grade of iron and that adequate gaseous reduction occurs in the upper part of the shaft to prevent the top-gas ratio from exceeding the stated values. If this control were lost, then direct reduction would be disastrous; but there is no reason to believe that any inherent limitation exists.

Fuel vs Gas Flow: With chemical requirements eliminated, the implication is that thermal requirements fix the coke rate. Indeed, this is the conclusion that many writers have reached. If Btu supply is the only limitation, however, the obvious solution is to increase the heat from hot blast and reduce the amount of coke at the tuyeres to give the same total heat balance in the hearth. Most furnaces have adequate stove capacity to move in this direction, but the furnace refuses to operate smoothly on a given burden with blast temperature above a critical level. This operating difficulty is the true limitation on coke rate, and it is necessary to look into the conditions governing gas flow and heat transfer in the furnace to seek an explanation.

Gas Flow in the Blast Furnace

A thoroughly quantitative treatment of gas flow through the blast-furnace shaft is impossible at the present time. Knowledge is inadequate to handle the nonuniform distribution of stock over the cross-section and the wide diversity in size and shape of solid particles. Changes in gas temperature and composition and in the horizontal cross-sectional area of the furnace along the path of gas flow are additional complications. Nevertheless, a semiquantitative approach gives a surprisingly useful insight into important operating variables. A variation of the Carman treatment[5-7] for fluid flow through a bed of granular solids will be used as the basis for this analysis. A more elaborate treatment is available,[8] but its use is not justified by the incomplete data at hand and would only make this presentation unnecessarily complicated.

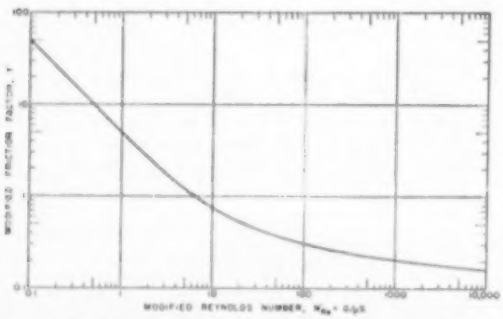

Fig. 1—Friction factor for beds of solids as a function of the modified Reynolds' number (after Carman[5]).

The friction loss or pressure drop incident to flow of a fluid through a bed of solids is given by Eq. 1, which is analagous to the familiar Fanning equation for pressure drop along a pipe

$$F = \frac{\Delta p}{\rho} = f' \frac{V_o^2}{g_c} \frac{LS}{\epsilon^3} \qquad [1]$$

$$\Delta p = f' \frac{G^2}{\rho g_c} \frac{LS}{\epsilon^3} \qquad [2]$$

where F is the mechanical energy lost in friction, ft-lb_{force} per lb_{mass}; Δp is pressure drop across the bed, lb_{force} per sq ft; ρ is the density of gas, lb_{mass} per cu ft; f' is the modified friction factor, dimensionless; V_o is the superficial mass velocity based on empty cross-section, ft per hr; g_c is a dimensional constant, 4.16×10^8 lb_{mass} — ft per lb_{force} — hr^2; L is the height of bed, ft; S is the specific surface of solids, sq ft per cu ft of bed volume; ϵ is the porosity, fraction of bed which is void volume; $G = V_o\rho$ is the superficial mass velocity, lb per hr per sq ft, based on empty column cross-section; and μ is the viscosity of gas, lb_{mass} per hr per ft. The modified friction factor, f', is a function of the flow conditions in the system as characterized by the modified Reynolds' number, $N'_{Re} = \frac{V_o\rho}{\mu S} = \frac{G}{\mu S}$. The relation between f' and N'_{Re} is shown in Fig. 1, from which f' may be read if the Reynolds' number is known, or from which an algebraic expression may be derived to represent a portion of the curve for a restricted range of flow conditions.

Numerical values for G, μ, and S typical of blast-furnace conditions must be estimated in order to proceed further. The only set of operating data known to the writer that approach being complete enough to be useful for this purpose are those published by Dobscha[9] for normal and size-beneficiated burdens. For purposes of approximation, the shaft of the furnace is imagined to be a straight cylinder of hearth diameter, namely, 27 ft 6 in. The gas composition and mass-flow rate are taken to be more nearly like the dry top gas than the bosh gas, giving a value of G of about 850 lb per hr per sq ft. The density and viscosity of the gas are assumed for an average temperature of 1500°F and gage pressure of 9 lb per sq in., giving $\mu = 0.12$ lb per hr per ft and $\rho = 0.033$ lb per cu ft. Estimation of porosity and specific surface is more difficult and is little more than a guess, since the screen intervals for the seive analyses of the ores were rather large and irregular and no sizes were given for other materials. It seems advisable to neglect the −100 mesh ore on the basis that such fine sizes either blow out with the flue dust or are attached to larger particles; otherwise, the specific surface values are determined almost entirely by the −100 mesh material. On the supposition that the bed is "fluffed" by the rising gases to a porosity of about 0.5 (possibly a low estimate), the average specific surface is taken to be about 300 sq ft per cu ft of bed for the normal burden and something over 200 for the beneficiated burden. In the central column of a blast furnace where large-sized material predominates, the specific surface is probably more nearly 50. Modified Reynolds' numbers for the blast furnace, therefore, are taken to lie roughly within the range from 20 to 150.

It is somewhat awkward that the range of modified Reynolds' numbers for the blast furnace lies in the curve portion of the correlation line of Fig. 1 in the transition between completely viscous flow at the left and fully developed turbulent flow at the right. However, the restricted portion of the curve for N'_{Re} between 20 and 200 can be reasonably approximated by the straight line described by Eq. 3

$$f' = 1.2_5 \, (N'_{Re})^{-0.3} . \qquad [3]$$

For modified Reynolds' numbers in the range from 100 to 1000, which might well be obtained with coarse burden materials, a better fit is given by $f' = 0.8(N'_{Re})^{-0.2}$.

Eq. 3 may now be substituted into Eq. 2 to obtain

$$\Delta p = 1.2_5 \left(\frac{G}{\mu S} \right)^{-0.3} \frac{G^2}{\rho g_c} \frac{LS}{\epsilon^3}$$

$$= 1.2_5 \frac{L}{g_c} \frac{G^{1.7} S^{1.3} \mu^{0.3}}{\rho \epsilon^3} . \qquad [4]$$

As a check, this solves to a pressure drop of about 20 psi for an 80 ft working height, S of 200 sq ft per cu ft, $\epsilon = 0.5$, and other values as noted earlier. This is of the right order of magnitude, fortuitously close to actual pressure drops, and offers some hope that qualitative predictions from Eq. 4 may be useful in spite of its oversimplification.

For further discussion, it may be easier to visualize charge materials in terms of average particle size rather than in terms of specific surface. For a bed of irregular particles, the following substitution can be made

$$S = \frac{6(1-\epsilon)}{D_s \psi}$$

where D_s is a *surface-average* particle size defined here, so that a bed of particles of uniform size D_s would have the same specific surface as the actual material.* Calculated D_s values corresponding to the specific surfaces estimated for Dobscha's charges are about 0.15 in. for the normal burden and 0.2 in. for the beneficiated burden, indicating the large effect of the small ore particles. Substituting this expression for S into Eq. 4 and rearranging gives

$$\Delta p = 13 \frac{L}{g_c} \mu^{0.3} \frac{[(1-\epsilon)/\psi]^{1.3}}{\epsilon^3} \frac{G^{1.7}}{\rho D_s^{1.3}} . \qquad [5]$$

Eq. 5 may be used to predict some interesting relationships among blast-furnace variables if Δp can be set equal to some limiting pressure drop above which the furnace will not operate smoothly. Inadequate blower capacity might fix the maximum blast pressure available, but more commonly the limitation is set by the onset of instability in the stock column. If gradually increasing pressure being applied at the tuyeres is imagined, the gas velocity will increase with pressure in accordance with Eq. 5 until the frictional forces tending to lift the pieces of stock exceed the total weight of the stock column. At this pressure, *aggregative fluidization*[8] will occur, and the gas will bubble upward through the charge. Further increase in pressure would increase the violence of the bubbling with the ejection of streamers of solids at the stock line. The limiting pressure drop for the beginning of aggregative fluidization is given[8] by

$$\Delta p = L(1-\epsilon) \, (\rho_s - \rho) \frac{g}{g_c} \qquad [6]$$

where ρ_s is the density of the solid particles, ρ the density of the gas, g the acceleration of gravity, and the other symbols are as defined† for Eq. 1. The gas density, ρ, may be neglected, since it is small compared with ρ_s, and the product $(1-\epsilon)\rho_s$ is the average bulk density of the bed, which may be represented by the symbol ρ'_s. Hence

$$\Delta p = L \, \rho'_s \, g/g_c . \qquad [7]$$

On the assumption that a blast furnace normally will be driven to the limit of pressure drop just short of excessive fluidization or instability of the stock column, the value for the limiting pressure drop given by Eq. 7 may be substituted into Eq. 5. The terms L/g_c appear on both sides and cancel out, and the constants 13 and g may be combined. The factor $\mu^{0.3}$ in Eq. 5 may be included with the constants, since it is not a very potent variable and gas viscosity is not subject to any deliberate control during furnace operation. Appropriate rearrangement leads to

$$\frac{[(1-\epsilon)/\psi]^{1.3}}{\rho'_s \epsilon^3} \frac{G^{1.7}}{\rho D_s^{1.3}} = \text{constant} \qquad [8]$$

or,

$$\frac{[(1-\epsilon)/\psi]^{1.3}}{\rho'_s \epsilon^3} \frac{\rho^{0.7} V_s^{1.7}}{D_s^{1.3}} = \text{constant.} \qquad [9]$$

Some limitations of Eq. 8 should be noted before it is applied to practice. Eq. 7 may be checked against Dobscha's study, in which the working height was 79.52 ft and the average bulk density of the charge was 66.6 and 69.5 lb per cu ft for the normal and beneficiated burdens, respectively. The limiting pressure drops calculate to 36.8 and 38.4 lb per sq in. compared with reported blast pressures of 20 and 22 lb per sq in. The actual pressure drop was well below that needed to fluidize the whole stock column. If a central core filled with coke only were assumed (bulk density = 30 lb per cu ft), the fluidizing pressure would be 16.6 lb per sq in. This suggests that the limiting pressure drop for normal filling may relate to a central core relatively rich in coke and lean in ore. For this portion of the stock column where large particle size predominates, it might be more appropriate to use the modified form of Eq. 3 for larger values of the modified Reynolds' number. This would change the numerical value of the constant in Eqs. 8 and 9, and the exponents 1.7, 1.3, and 0.7 would become 1.8, 1.2, and 0.8, respectively. The latter values fit the data reviewed by Joseph[1] in which pressure drop in a bed of solids varied with the 1.8 power of the flow rate and inversely with the 1.2 power of the particle diameter. Since it is impossible at present to account exactly for actual distribution of charge within a blast furnace, Eq. 8 is intended only to show the interrelations among the variables and is not claimed to be fully quantitative.

Comparison of Equations with Practice: The terms in Eqs. 8 and 9 now will be identified with operating

* The specific surface of a sphere is $6/D$. The *sphericity factor*, ψ, corrects for the greater surface area per unit volume of nonspherical particles: ψ = (Area of sphere of same volume as a particle)/(Surface area of the particle). Values of ψ range from 0 to 1. The factor $(1-\epsilon)$ is the fraction of bed volume occupied by particles.
For a distribution of sizes of particles of the same shape and density, $1/D_s = \Sigma (f_1/D_1)$, where the right-hand member represents the sum of all the quotients obtained by dividing the mass fraction of each screen size by the average screen aperature for the fraction. If particles of different densities are involved, volume fractions rather than mass fractions must be used.

† Note that g and g_c have the same numerical value if expressed in consistent units, and that Eq. 8 gives Δp in lb_{force} per sq ft if English units are employed.

variables, and the equations will be tested to see if they give any useful predictions. The left-hand fraction contains bed characteristics only, while the more important operating variables have been grouped in the right-hand fraction.

The G term expresses production rate rather closely. The pounds of gas per hour per square foot of shaft area is nearly (though not exactly) proportional to the pounds of coke burned per hour in a given hearth area. Since gasification of coke above the tuyeres adds to gas weight and since production rate varies with coke per ton as well as with wind rate, some judgment must be used in comparing G with production rate when substantial changes in practice are involved. In Eq. 9, $V_s \rho$ replaces G. This form of the equation is useful if a comparison of linear velocities of gas in the throat of the furnace is of interest in connection with flue-dust losses.

The density of the gas, ρ, varies with pressure, temperature, and composition. Pressure is the only one of these factors which is amenable to independent control in blast-furnace practice, and then only by operating with increased top pressure. An increase of top pressure to 10 lb per sq in. serves to increase the average static pressure and, therefore, the average density of the gas by about 30 pct. Note that the pressure drop through the furnace may decrease with an increase in operating pressure, and that absolute rather than gage pressure is to be used in calculating gas density.

The average particle size, D_s, depends on the size distribution for the entire charge but is determined most strongly by the smaller-sized materials. The average size varies inversely with specific surface, sq ft per cu ft. Most of the surface is accounted for by the ore particles under ¼ in. in size, but the coke and stone are important in that they occupy a large fraction of the volume of the furnace while contributing comparatively little surface area. Thus, the large material increases D_s about in proportion to its volume in the furnace relative to a given ore charge.

In the first fraction on the left side of Eq. 8, the term $[(1-\epsilon)/\psi]^{1.8}$ has only a weak influence. The effective sphericity of a granular bed tends to increase as the packing density $(1-\epsilon)$ increases (porosity decreases), and the changes are partly compensating. The factor $1/\epsilon^3$ is a potent one. For porosity values of 0.50, 0.45, 0.40, and 0.35, the respective values of $1/\epsilon^3$ are 8, 11, 15.6, and 23.4; the bed resistance doubles for a 20 pct decrease in porosity from 0.5 to 0.4 and triples for a 30 pct decrease to 0.35. Joseph[1] gave a good review of the adverse effect on porosity of mixed particle sizes. As he pointed out, small-sized particles inevitably cause a high resistance to gas flow (D_s in Eq. 8), but their effect is compounded if they are mixed with larger particles to give a minimum porosity as well. Exactly the same principle is expressed in somewhat more quantitative terms in Eq. 8. Joseph made a strong case for size segregation of ore and charging to give a small range of sizes in any given layer, but only a few blast-furnace operations have made practical use of this knowledge.

In the derivation of Eq. 8, it has been assumed that a blast furnace normally will be driven to the limit of pressure drop just short of excessive fluidization or instability of the stock column, so the overall value of Eq. 8 will remain constant. A change in one factor, therefore, requires or permits a compensating change in another. If D_s or ρ or ϵ is increased in some way, then the driving rate, G, may be increased, but not in exact proportion. Likewise, an increase in the bulk density, ρ'_b, of the charge because of an increase in burden ratio (or perhaps because of placing a higher proportion of ore in the central core), would permit a faster driving rate if such an increase in charge density could be made with sized ore so as not to affect porosity or average particle size adversely. A few specific examples will be given for illustration.

Size Beneficiation: Dobscha[9] has reported a careful operating comparison of furnace performance on a normal Lake ore burden and on a beneficiated burden constituted of coarse ores and concentrates with much of the −⅜ in. material screened out, plus 20 pct sinter. Neglecting small changes in gas density, charge bulk density, and unknown, but probably not very great, changes in charge porosity, Eq. 8 predicts that

$$\frac{G_1^{1.7}}{D_1^{1.8}} = \frac{G_2^{1.7}}{D_2^{1.8}}; \left(\frac{G_2}{G_1}\right)^{1.7} = \left(\frac{D_2}{D_1}\right)^{1.8}; \frac{G_2}{G_1} = \left(\frac{D_2}{D_1}\right)^{0.77}.$$

Estimated average particle sizes were given earlier as 0.15 in. for the normal burden and 0.20 for the sized burden, so

$$\frac{G_2}{G_1} = \left(\frac{0.20}{0.15}\right)^{0.77} = 1.25 \cdot$$

This predicts a 25 pct increase in production rate as against 21.2 pct observed. However, the observed gain was not obtained from a higher gas flow rate (wind rate increased 2.4 pct and dry top-gas flow rate decreased 1.2 pct on a mass basis), but from a lower coke rate! Some discussion is necessary to salvage the theory.

This discrepancy points up a serious problem in attempting to extend correlations based on comparatively uniform beds to the complex distribution that exists in a blast furnace, without adequate experimental basis. It is quite possible that the very smallest ore particles should be disregarded in computing specific surface, either because they are attached to, and behave as, part of larger lumps or because they dance in the gas stream in interstices and do not add to flow resistance as strongly as the specific surface implies. Only the −100 mesh material was disregarded in the writer's estimations. The more likely possibility is that gas flow is channeled preferentially in the central core where larger sizes predominate, as noted in the first paragraph following Eq. 8. If average size in the central core is determining, then an increase in burden relative to coke would decrease this particle size appreciably by putting more fine material in the center of the shaft.

Perhaps it was only because of the increase in average size of the ore in the beneficiated burden that an increase in burden ratio or decrease in coke rate was possible; one change compensated for another to hold $D_2 = D_1$ so G_2 could equal G_1. It happens that Dobscha's beneficiated charge was improved chemically as well as physically.[10] There was less oxygen to be removed from ore, less gangue to flux, and less stone to be calcined and to contribute CO_2 to the stack gases. The lower chemical and thermal requirements called for less coke. It was possible for the operator to capitalize on this potential saving and maintain the same wind rate with less coke per unit of burden because of adequate porosity. Without the size improvement of the ore, but only chemical improvement, it might well have been necessary to blow at a slower rate or to charge more coke to maintain permeability of the stock column.

It is significant that full advantage apparently could not be taken of the lower thermal requirement of the beneficiated burden, because the blast temperature was only 916°F compared with 969°F for the normal ores. Presumably a higher blast temperature would have been used to increase production rate still more if the furnace would have taken it. This point will be considered again in a later section.

It is concluded that application of Eq. 8 to Dobscha's experiments does not disprove the equation but highlights the need for further experimentation to learn how to utilize the basic theory under the complex conditions in a blast furnace.

High Top-Pressure Operation: The flow equation predicts that an increase in production rate should be obtained by operating a furnace at higher than normal pressure simply through the effect on gas density without consideration of chemical effects on solution loss or reaction rates. The data reported by Slater[13] may be used to check this prediction. Assuming no change in the character of the burden, Eq. 8 gives

$$\frac{G_1^{1.7}}{\rho_1} = \frac{G_2^{1.7}}{\rho_2}\,;\ \left(\frac{G_2}{G_1}\right) = \left(\frac{\rho_2}{\rho_1}\right)^{0.59}\cdot$$

Since gas density is directly proportional to pressure, p_2/p_1 may be substituted for ρ_2/ρ_1. Average pressures for Slater's operation were $(20+2)/2 = 11$ psig or 25.7 psia for normal operation, and $(26+10)/2 = 18$ psig or 32.7 psia for high top-pressure conditions.

$$\frac{G_2}{G_1} = \left(\frac{32.7}{25.7}\right)^{0.59} = 1.15\cdot$$

This predicts a 15 pct increase in production rate when blowing with maximum wind, which is in the middle of the range 11 to 20 pct reported by Slater. Improvement in flue dust is best seen from the relation $G = V_s\rho$. If G is held constant, an increase in ρ (by pressure) gives a corresponding decrease in linear velocity of the gas at the stockline. Even with full advantage taken of the possibility of increasing wind rate, Eq. 9 predicts a decrease of 9 pct in top-gas velocity for pressure operation. Some saving in coke at less than maximum wind rate can be expected partly on the basis of lower linear velocity, longer time of contact, and hence more efficient usage of reducing gas (if, indeed, a lower CO/CO_2 ratio is noted). It is also obvious that an increase in ρ permits a lower D_c for the same gas flow rate, so it may be possible to operate with less coke required simply to give permeability in the stock column and to replace the thermal deficiency by hot blast heat. Eq. 8 also predicts that, by increasing gas density, it should be possible to maintain a given production rate, G, with finer ores (smaller D_c). All-in-all, the equation adequately explains the observations with respect to high top-pressure operation.

Summary of Relations to Limiting Factors: This section has made the point that the pressure drop through the blast furnace, and hence the blast pressure, is limited by the conditions under which aggregative fluidization or stock-column instability would set in. In order to attain an economically attractive wind rate, the size and shape of the charge particles and the porosity of the bed must provide adequate permeability for gas flow. It appears that the central core of large-sized material may be essential to provide adequate flow for commercial rates of operation. An appreciable portion of the coke requirement may be needed simply to ensure the necessary large size and high permeability of this portion of the stock column. If this is true, then such coke represents an economic cost of high production rate. The presence of this coke in the charge restricts the use that can be made of blast heat without losing control of hearth temperature. Gas-flow requirements therefore may limit the minimum coke rate consistent with a reasonable production rate.

An inference to be drawn from the principles just outlined is that coarse ores should permit lower coke consumption with higher blast temperatures, and this fits practical experience. There is reason to expect a substantial saving in coke and an increase in production rate with the use of agglomerates of relatively large and uniform size produced from concentrates, and such savings will be credits to apply against the cost of beneficiation. The adverse effect of large lump size on rates of heating and reduction will be dealt with in a later section.

While the foregoing treatment has provided a possible explanation of operating limitations to the greater use of hot blast heat, the explanation is a long-range one and does not account well for difficulties that might develop quickly and without any reduction in coke charge. Critical conditions under this theory should be indicated by rolling, blowing through, and excessive flue dust, rather than by hanging. It is believed that other factors may be controlling under certain circumstances, and these will now be explored.

Flooding Conditions

Elliott, Buchanan, and Wagstaff[12] made the ingenious observation that conditions in the fusion zone of a blast furnace are similar to those in a countercurrent vapor-liquid extraction tower, in which flooding occurs if either the gas or liquid flow rate is increased beyond a critical condition. They demonstrated by a model that the flooding criterion is the same when the liquid is formed by melting within the column as for normal feeding from the top. Calculations based on actual blast-furnace practice, as nearly as the pertinent factors could be estimated, showed that normal operating conditions are probably not very far below the critical condition for flooding. Their Fig. 9 is reproduced for convenient reference as Fig. 2 of this paper, with a few symbols* changed to conform with those used here.

* The symbol a was incorrectly defined in the title of Fig. 9 of ref. 12; it should have been defined as specific surface, S, as used in this paper. The symbol a was used correctly to denote specific surface in the text and calculations.

Elliott and his coworkers gave such a clear and thorough discussion of the meaning of the flooding parameters and their relation to blast-furnace operations that there is no need to do more than draw some comparisons with the overall flow-resistance principles developed herein. They called the two flooding parameters the "fluid ratio" and the "hanging factor," as defined on the axes of Fig 2. The meaning of the symbols is the same as given following Eq. 1, but with subscripts G relating to gas and L to liquid (slag); V_s refers to superficial gas velocity and μ is the slag viscosity in centipoises. The fluid ratio has about the same numerical value for both iron and slag, and its range of variation is so small as to be unimportant. The hanging factor therefore is the controlling one. The hanging factor calculated for slag is substantially greater than that for iron because of the lower density and higher vis-

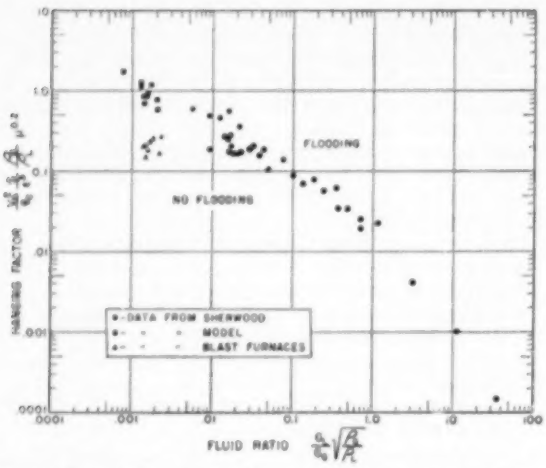

Fig. 2—Correlation of flooding of packed columns (Elliott, Buchanan, and Wagstaff[2]).

cosity of the slag. It is assumed, therefore, that slag will be the liquid involved if flooding occurs.

The properties included in the hanging factor are, in part, the same ones that are important to overall resistance to gas flow. There is an important distinction, however. The danger of flooding occurs only at the fusion zone in the bosh where the slag is forming. The specific surface and porosity are, therefore, those of the coke alone rather than a composite for the charge. The gas density and mass flow rate are reckoned for bosh gas at appropriate temperature and pressure and for bosh diameter, rather than for average conditions higher in the shaft.

The consequence of flooding would be that slag would hold up in the interstices of the coke and ultimately be regurgitated to a higher level in the furnace where it would freeze, blocking passages for gas flow. The effect would be self-propagating and would rapidly lead to hanging and a sharp rise in blast pressure and a decrease in wind rate. The hanging factor predicts an increasing tendency toward flooding with high gas velocity, small coke size and low porosity, and high slag viscosity.

The flooding of coke by slag offers a very convincing explanation of certain kinds of furnace difficulties, especially with small weak coke or very viscous slags. It does not, however, offer a complete theory for all limitations on production rate or coke rate. In fact, only the characteristics of the coke are involved, but not the burden ratio, so it is hard to see how flooding can account in any way for a restriction on minimum coke rate. On the contrary, a low coke charge would be expected to result in a less limey and more fluid bosh slag, if anything, and less tendency toward flooding. The fluid ratio might be increased slightly by decreased coke consumption (higher ratio of slag flow to gas flow), but this term has only weak influence. There is no reason to think that blast temperature would have any continuing influence on flooding, although a sudden change in blast temperature might conceivably have a transient effect. Finally, the calculated hanging factors for actual operation are lower than the critical values indicated in Fig. 2 by about a factor of 4. In spite of the acknowledged uncertainties in estimating conditions inside the furnace, it seems unlikely that the margin of error is quite 400 pct and that furnaces normally work at the ragged edge of flooding.

It is proposed that both overall flow resistance and flooding tendencies are potential limits to wind rate. Normally the overall resistance becomes controlling first; but with poor coke or unusually viscous slags, flooding may occur first and become the limiting factor. Neither theory is entirely adequate to explain some peculiarities of furnace behavior. In particular, both would predict an advantage from oxygen enrichment, which reduces gas volume for a given rate of coke combustion and should promote smoother operation, but experience does not seem to bear this out. Still another limitation may exist, and this is inherent in the furnace lines relative to the location of the fusion zone.

Location of the Fusion Zone

To attack this phase of furnace operation, the heat balance in the hearth and heat transfer from rising hot gases to descending stock must be considered. It is assumed that a nearly constant temperature level must be maintained in the "hearth zone" to produce iron of specified analysis. The upper boundary of the "hearth zone" is taken to be a short distance above the tuyeres, but it is not necessarily a horizontal plane. The temperature is maintained constant by a balance between the heat released by combustion of coke plus hot blast heat and the heat consumed by hearth reactions, cooling losses, and heat carried out in the gas. Gases leaving this hearth zone are assumed to leave at a constant temperature level and to carry sensible heat content in proportion to their volume and volumetric heat content at the hearth temperature.

The important consideration in the heat balance in the bosh is the volume of gas rising upward relative to the total quantity of stock coming down in unit time. Suppose the relative quantity of gas to be decreased. This could occur from less coke burned at the tuyeres per ton of iron (since the gas is formed by burning coke) or from oxygen enrichment (decreasing the nitrogen content of the gas). The gas carries less heat upward per unit of descending stock, even though the temperature of the gas is initially at the normal level. The gas cools more rapidly because of its lower heat content relative to the heat capacity of the stock. The level in the bosh at which the fusion-zone temperature occurs therefore must fall. Furthermore, the temperature levels all the way up the stack must fall, and the top-gas temperature will be lower as well.

The lines of a blast furnace are designed to allow an increase in volume proportional to the thermal expansion of the stock as it settles through the in-wall section. The lines contract again in the bosh to compensate for contraction during fusion of the iron and slag and from some consumption of coke by reaction with CO_2. If the fusion zone is lowered drastically below the level for which the furnace was designed, the stock is still being heated and expanding when it reaches the constricting lines of the bosh. If the furnace offers no room for the necessary expansion, the charge must be compacted, with resultant bridging, hanging, and increase in blast pressure. The furnace must be checked.

Lowering of the fusion zone offers a rational explanation of why a furnace will not operate smoothly with too high a blast temperature and of similar difficulties that the writer understands have been encountered with oxygen enrichment. Other things being equal, a higher blast temperature demands that less coke be burned at the tuyeres per ton of iron in order to maintain the hearth tempera-

ture in balance. This results in less gas per unit of charge. Oxygen enrichment of itself generates no more heat, because the heat is released by combustion of coke with oxygen in any event. The reduced nitrogen content, however, carries less heat away from the hearth zone. This in turn changes the hearth heat balance in the direction of less coke, which further reduces the relative volume of rising gas. Not only is the fusion zone lowered, but the top-gas temperature falls also.

There is considerable confirming evidence for a belief that any change in furnace performance that leads to the formation at the tuyeres of less gas per unit of charge will bring the fusion zone closer to the tuyeres. The general trend in furnace design over the years has been in the direction of shorter boshes, concurrent with increased efficiency of operation. Fulton's experience[13] provides a striking example of this. While an existing furnace design may permit a considerable degree of improvement, a point ultimately may be reached where the bosh lines restrict any further saving in coke. The fact that the blast temperature was lower for Dobscha's beneficiated burden than for the normal charge suggests that the limit on coke rate may have been reached for those furnaces. It seems almost certain that further evolution in furnace lines to still shorter boshes, and probably to lower stocklines to prevent moisture condensation in downcomers, will be necessary to realize the ultimate advantages to be gained from coarser charge materials or oxygen enrichment. Unfortunately, an exact basis for engineering design is lacking.

Heating and Reduction of Ore

The preceding discussion has emphasized that charge materials ideally should be large in size to give minimum resistance to gas flow. Large particles, however, require more time to be heated and reduced, so it is obvious that there must be some compromise on size. A brief treatment will be given to show that the limitation on maximum size is reduction rather than heating and to try to bracket the optimum size for ore.

Heat Transfer: A usefully complete treatment of heat flow to, and within, ore particles would greatly extend this already long paper, so the reader is referred to ref. 6 or to specialized texts on heat transfer. The heat-transfer coefficient for heat flow from a hot gas to the surface of particles in a bed may be correlated with the thermal properties of the gas and with the flow conditions as characterized by the modified Reynolds' number (DG/μ). Unfortunately, the several investigations that have been made are not entirely in agreement, and they have dealt with much simpler particle shapes and bed characteristics than exist in a blast furnace. A further complication arises from the fact that the blast furnace apparently operates in the transition range of Reynolds' numbers where the correlation changes continuously with the flow characteristics. It seems inappropriate to cite any one relation as being quantitatively applicable to the blast furnace, but calculations of the heat-transfer coefficient from several likely possibilities have yielded results nearly enough in agreement for the present purpose.

The surface heat-transfer coefficient can be utilized in one of the standard solutions for unsteady-state heat transfer to approximate the temperature within the particle as a function of time. The density, specific heat, and thermal conductivity of the solid must be known, and some guesses have been made of these properties for partly reduced ore particles at a temperature of about 1500°F (800°C). In order to permit direct comparison with reducibility data, a calculation was made of the time required for 90 pct of ultimate heat absorption by cubic particles. The time varies inversely with the square of the edge length of the cube. The results are shown as a band in Fig. 3. It is emphasized that this range is only a rough estimate at best.

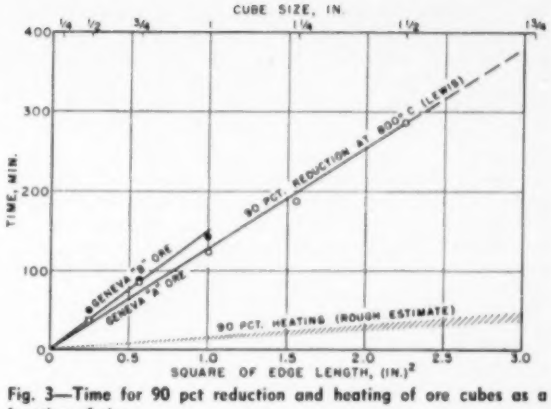

Fig. 3—Time for 90 pct reduction and heating of ore cubes as a function of size.

Reducibility: The reduction of ore particles is governed by diffusion processes that are analogous to thermal diffusion. The internal mechanism is complex,[14] and the controlling factor in iron-ore reduction is probably gaseous diffusion and convection in pores originally present and created during the process itself, rather than solid diffusion. The practical problem is complicated further by differences in the composition and structure of the iron-bearing burden components and by the irregular shapes of natural particles. It might be expected that the basic similarity of mass diffusion in ore reduction to thermal diffusion would be masked by the superimposed variables of structure and shape. It is surprising to find that Lewis' data[15] on the effect of size on time for 90 pct reduction of cubes of Geneva iron ore follow almost exactly the D^2 relation expected of simple diffusion. Lewis's data are replotted in this way in Fig. 3.

The reduction times in Fig. 3 are considerably longer than the heating times for cubes of equal size. The Geneva ores were hematites of intermediate reducibility. Reduction times for other hematites and for nodules and pellets would be expected to group around the plotted values, and magnetites and sinters would take considerably longer to reduce. It seems safe to conclude that all ores and sinters can be heated to a given temperature faster than they can be reduced at that temperature. Since reduction by carbon monoxide is exothermic, the heat of reaction would raise the temperature at the reacting surface rather than lowering it. The rate of reduction will not be limited seriously by the rate of heat supply. Maximum size of ore particles therefore should be determined by reducibility requirements rather than by heat transfer.*

* From the reverse viewpoint of recovery of sensible heat from the gases by the stock, a large particle size would adversely affect the heat-transfer coefficient per unit of bed volume. However, the large size would also decrease pressure drop and permit a taller column, if necessary, for good thermal efficiency. It is believed to be so unlikely that heat-transfer requirements will be a limiting factor on ore particle size that a more quantitative analysis is unnecessary.

It must be understood that Fig. 3 is useful only for a crude comparison of the relative times for heat-

ing and for reduction under similar conditions of immersion of ore cubes in gas of constant temperature or constant reducing power. The data plotted can not be used to estimate any reduction time under actual blast-furnace counter-flow conditions. Under present knowledge, the only real test for a limiting size for adequate reducibility of a given charge material is performance in the blast furnace. If an increase in size causes an increase in top-gas CO/CO_2 ratio and coke consumption, then the limiting size has been passed. Laboratory tests can help only to extend predictions from limited performance tests to other sizes or materials. Since a certain amount of "direct reduction" is desirable anyway, and reduction time varies with the square of the particle size as well as with inherent properties of the material, great caution must be used in judging the virtues of a blast-furnace feed material on a laboratory reducibility index alone. Quality should be defined in terms of use, and there is no point in increasing costs to obtain quality much better than is good enough for the intended application.

Optimum Size for Blast-Furnace Feed

An increase in particle size above that prevalent in present American blast-furnace ores would lead to higher permeability to gas flow in the ore-rich portions of the stock column. This would permit an increase in production rate through an increase in wind rate, decrease in coke rate, or both, depending on the original practice used as a basis for comparison. Most of these advantages would be gained by eliminating $-\frac{1}{2}$ in. particles from the burden, and the maximum improvement would be expected with ore of about the same size as the coke, or about 2 or 3 in. in size.

From the standpoint of reducibility, there is a limited amount of operating and laboratory data which can be used to estimate maximum ore size.[16-18] Fairly porous natural hematites and limonites and hematitic pellets or nodules probably have adequate reducibility in sizes up to 2 in. The slower reducing magnetites and hard sinters seem to require sizing down to $\frac{1}{2}$ to $\frac{3}{4}$ in.[18]

It is concluded that the optimum size for a screened ore or for an agglomerate produced from fines or concentrates is probably within the range from $\frac{1}{2}$ to 2 in., with the upper side of the range favored for materials of good reducibility.

Summary and Conclusions

An effort has been made through the application of engineering and scientific principles to develop a complete analysis of the factors that restrict the production rate and coke economy of the blast furnace in as quantitative terms as present knowledge permits. The daily productivity of a blast furnace depends on the rate at which blast can be supplied without upsetting smooth operation of the furnace and on the pounds of coke that must be burned at the tuyeres to produce a ton of iron. It has been shown that the limiting coke rate under present blast-furnace practice is not determined by chemical or thermal requirements directly, but by considerations of gas flow and stock descent. The wind rate is also circumscribed by resistance to the flow of gases in the stock column, so the two main variables are interrelated.

It is believed that under most circumstances the maximum rate of gas flow through the shaft of the furnace is limited by an overall pressure drop above which fluidization of the bed becomes excessive, leading to blowing through, rolling, and abnormal flue-dust loss. The permissible gas-flow rate is increased by increasing average particle size, increasing gas density (top pressure), and increasing porosity of the bed, in accordance with Eq. 8. The removal of fines from ore and size segregation before charging are pointed out as effective ways of increasing the permeability of the stock column to permit an increase in production rate.

A charge bearing a large proportion of fine ore seems to require a central core of large-sized material, primarily coke, to give the permeability necessary to permit a high enough production rate for best overall costs. The gases rise preferentially through this path of least resistance where they come in contact with relatively little ore, so that the gas is not used to best advantage in heating and reducing the charge. The coke in the central core is used inefficiently to increase production rate at the expense of coke rate. This coke is chemically and thermally unnecessary, and its combustion at the tuyeres not only decreases the production rate but also limits the use that can be made of hot blast temperature without upsetting the hearth heat balance. Decreasing the proportion of fines in the ore improves the overall permeability of the stock column and reduces the need for coke simply to vent the gas. A lower coke consumption follows from improved efficiency and a greater use of hot blast heat until some other limiting factor becomes effective. For a given wind rate, increased tonnage is automatic if more iron is produced per unit of coke burned at the tuyeres.

With small weak coke or with unusually viscous bosh slags, the flooding of the coke column in the bosh with slag may restrict the wind rate to a lower value than the limit placed by overall resistance of the stock column. Flooding leads to hanging and related difficulties. This flooding tendency accounts for a limit on wind rate, but does not seem to be related directly to coke rate or blast temperature.

A final limitation on coke rate and blast temperature may be attributed to the lines of a particular furnace. A decrease in coke per unit of burden, or increase in burden ratio, leads to a lower volume of gases leaving the hearth zone per unit weight of descending stock. Oxygen enrichment of the blast would have the same effect by reducing the nitrogen content of the bosh gas. This lowers the position of the fusion zone in the furnace. If the stock has not reached the fusion temperature but is still expanding when it reaches the constricting lines of the bosh, bridging and hanging will occur. The lines of a furnace therefore may dictate that the coke rate and blast temperature must be adjusted to give at least the limiting volume of bosh gas per unit of charge. The ultimate gain from beneficiated burdens or oxygen enrichment of the blast probably will not be attained without some modification of blast-furnace lines in the direction of shorter boshes and lower stocklines.

In principle there should be an optimum compromise on an ore size large enough for good gas flow and coke rate but small enough for sufficiently rapid heating and reduction. It appears that reducibility rather than heat flow limits the top size. A preliminary estimate places the optimum size in the range from $\frac{1}{2}$ to 2 in. depending on the reducibility of the burden material.

With the developing trend toward greater use of prepared agglomerates, there is a remarkable oppor-

tunity to design the feed and the process to fit one another to best advantage in contrast to forcing the process to take what is conveniently available. There is every reason to expect that substantial additional improvements can be made in blast-furnace productivity and fuel costs, which will repay at least a portion of the cost of raw-material beneficiation. The changes in burden characteristics should be in the direction of larger size and greater uniformity of size. Some thought should be given to variations in charging procedure to work toward more uniform distribution of materials over the stock line as ore size approaches coke size. There is no reason to be complacent that the ultimate saving has been reached until blast temperature has been pushed up to the limit of economically reasonable stove capacity and the limitation on coke consumption has been moved back to fuel requirements, where it belongs.

It is believed that this analysis accounts in a broad way for the major variables and limitations in blast-furnace operation relating to production rate, coke requirement, and properties of charge materials, aside from the determination of the final composition of the iron and slag. While most of the conclusions taken individually are not new, it is hoped that this will give a more comprehensive and more nearly quantitative picture of the whole problem than heretofore has been available. A truly quantitative analysis will not be possible until adequate engineering treatments are developed for gas flow, heat transfer, stock movement, and reduction rates in a column with the heterogeneous distribution of materials by both size and kind that occurs in the blast furnace, or until the stock column has been made more homogeneous. Much research and development work is still needed, some of which is best attempted in the laboratory while other aspects require full-scale operation. For the present, it must be recognized that this paper of necessity is based on extended extrapolations. Conclusions or predictions should be used with caution, checked constantly against practice, and modified as better information becomes available.

Appendix

An analysis similar to the one given here was published recently by Ergun.[19] Discussion and communications from J. B. Wagstaff and H. H. Lowrie incident to the presentation of this paper have indicated that a brief comparison of the two treatments will be useful.

The present paper, which has evolved over a period of three years, is intended primarily to demonstrate method of attack and to show interrelationships among variables because quantitatively exact predictions are not believed possible at the present time for reasons stated before. It seemed best for this purpose to base the treatment on simple and generally accepted engineering correlations that are available in standard texts or reference works, and the Carman treatment for fluid flow through granular beds was selected to relate pressure drop to blast-furnace variables. Ergun based his analysis on his own correlation,[20] which he believes to be more valid on theoretical grounds than earlier treatments. Ergun's equations may be put into the form of Eq. 2 of this paper. His D_p is a "geometric" particle size that is defined to be equal to $D_s\psi$ in the relation below the author's Eq. 4. Taking proper account of definitions and units and the incorporation of a factor of 6 in the numerical constants, Ergun's relation (Eq. 14b of ref. 20) that is equivalent to Fig. 2 becomes

$$f' = 0.292 + 4.17/N'_{Re}. \qquad [10]$$

If Eq. 10 is plotted on Fig. 2, the two curves are nearly identical in the range of modified Reynolds' number from 10 to 100, which covers most of the region that was taken to be applicable to the blast furnace in this paper. There is in point of fact, then, no serious disagreement in the relation of friction loss to flow conditions within the region used in this paper, although the exponential approximation adopted here for mathematical convenience, Eq. 3, is quite different in appearance from Eq. 10.

The major discrepancy between the two analyses lies in the estimations made of the average particle size and void volume in the stock column. Ergun's selections, to be discussed in greater detail below, lead to values of N'_{Re} (as defined here) of the order of 1500 or higher. In this region, Ergun's equation (Eq. 10) lies above the line of Fig. 2 and differs from most other correlations in that f' is virtually independent of N'_{Re}. His analysis[19] of pressure drop in the blast furnace is based on what is equivalent to substituting $f' = 0.30$ in Eq. 2 in place of Eq. 3. If proper account is taken of definitions, units of measurement, and combining of constants, this is identical to Eq. 2 of ref. 19. From this point onward, the forms of the equations and the mathematical treatments are different, but the points of similarity and difference should be obvious to those who have sufficient interest to compare them in detail.

Ergun used an estimated void fraction, ϵ, of 0.29 for blast furnaces, based on a "study of available information on bulk weight and density of raw materials, their relative amounts, and apparent specific gravities of blast furnace charges . . ." The present writer attempted to estimate void fractions for Dobscha's burdens from the quoted bulk densities and reasonable guesses as to apparent densities of particles, since there were no experimental data to go on. The resulting estimates of ϵ were 0.38 and 0.40 for the regular and beneficiated ores, respectively, and about 0.50 for coke and 0.39 for stone. For the overall averages for the charges, the calculated ϵ's were 0.45 and 0.46 for the regular and beneficiated burdens, assuming no size degradation and layered charges without mixing. A somewhat higher value of $\epsilon = 0.5$ was used in this paper in the belief that the bed is probably expanded to some extent by the rising gases, as indicated by the fact that blast pressure tends to be higher than normal for a given wind volume for a short time after the wind has been taken off, as in plugging the taphole.

Ergun[19] stated: "Some fines adhere to coarse particles, while most are forced by gas into pockets between coarse pieces, resulting in the formation of channels. Therefore, there is a lower limit to the size of the fractions to be included . . ." in the calculation of average particle size. In Ergun's calculations "only +8 mesh-size fractions have been included." No specific reason was given for the cutoff at 8 mesh rather than some other size, but this was probably found by trial and error to lead to calculated pressure drops of the right order of magnitude in conjunction with the selected value of $\epsilon = 0.29$. It may be noted also that Ergun made no adjustment for particle shape in estimating D_p from screen-analysis data; otherwise, the methods of calculation are the same as used here. In this paper it was recognized also that extremely small particles had to be ignored to obtain reasonable average sizes. The cutoff was arbitrarily put at 100 mesh, and this gave reasonable results for the void fraction of 0.50 assumed. Ergun's selection of 8 mesh as the minimum size to be considered lead to average particle sizes about 10 times those calculated here from a 100 mesh cutoff size.

The differences in particle size and void fraction estimated to be typical of blast-furnace burdens in the two papers under discussion lead to differences in the modified Reynolds' number by a factor of 10 to 20, which in turn placed the calculations on different regions of the correlation curve of Fig. 2 or Eq. 10. This, of course, leads to quantitatively different results. The differences are largely matters of judgment which can be settled only by more exact information on blast-furnace operation. It also remains to be shown whether or not correlations for gas flow in randomly packed beds may

be applied without modification for layered and channeled charges such as occur in the blast furnace.

For the present, this writer feels that these two independent analyses of the same problem should be considered as complementary rather than competitive and that both are open to substantial revision with increase in knowledge of blast-furnace engineering.

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Technical Note

Evidence for Order in the Mn-Mo Sigma Phase

by B. F. Decker, R. M. Waterstrat, and J. S. Kasper

THE large difference in the X-ray scattering factors for molybdenum and manganese makes the σ phase in the Mn-Mo system an interesting one to study with respect to ordering of the atoms. Such a study has been carried out using data obtained from a powder sample, and the resulting evidence for ordering in this compound is presented here.

It is not possible, of course, to obtain all the details of structure for a σ phase from powder data only. Most difficult of all would be the determination of precise atomic parameters. There is reason to believe, however, that the atomic parameters do not vary greatly in the different σ's and that the precise values obtained from one σ structure would then allow decisions as to the ordering of atoms to be made when there is a large difference in scattering factors. In particular, the powder intensities for Mn-Mo σ are for the most part in good agreement with those observed for Co-Cr, for which single crystal studies have been made by the authors.[1] There are a few marked discrepancies in the intensity comparison of the two σ's, and these cannot be removed by adjustment of the parameters. They are interpreted as resulting from a more pronounced effect of ordering of the atoms.

Accordingly, the parameters of Co-Cr σ[1] have been used for the calculations of the Mn-Mo intensities. For Co-Cr, the space group is D_{4h}^{14}—$P4/mnm$ with atoms in the following positions: I—2 in (b); II—4 in (f), with x=0.1031; III—8 in (i), with x=0.3713 and y=0.0370; IV—8 in (i), with x=0.5662 and y=0.2396; and V—8 in (j), with x=0.3160* and y=0.250. The unit cell is tetragonal, with a_o=9.10Å and c_o=4.74Å for the Mn-Mo σ phase studied here.

* Actually, it is believed that for position V, the value of x is either 0.3063 or 0.3256, depending on which kind of atom occupies the position. The data for the present report do show clearly that associating Mo atoms with 0.3063 and Mn with 0.3256 is not good. The reverse association is good, but the data are not sufficiently accurate to distinguish between such a model, and one in which both Mn and Mo atoms are placed in the average position, with parameter 0.3160.

The material was prepared by a method described elsewhere.[2] Chemical analysis yielded 63.7 atomic

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Table I. Photographic Data for Mn-Mo σ Phase (Cr Kα)

hkl	d,A	$I_{obs.}$	Ordered $I_{calc.}$	Disordered $I_{calc.}$
110	—	0	0	0
200	—	0	0	0
101	—	0	7	3
210	—	0	0	0
111	—	0	2	0
220	—	0	1	0
211	—	0	0	0
310	—	0	4	1
221	—	0	3	1
301	—	0	1	0
320	—	0	6	0
311	2.47	18	18	7
002	2.38	20	19	23
400	—	0	3	0
321	—	0	1	1
112	—	0	1	3
410	2.22	80	83	87
330	2.15	35	37	36
202	2.11	40	40	35
212	2.06	75	72	66
420	—	0	0	0
411	2.01	100	96	100
331	1.959	50	45	50
222	1.915	12	8	11
421	—	0	0	1
312	1.835	12	7	11
430	—	0	0	0
510	—	0	0	1
322	—	0	0	3
431	—	0	1	3
501	—	0	1	0
520	—	0	0	0
511	—	0	1	3
402	—	0	1	0
412	—	0	0	0
440	—	0	0	0
521	—	0	1	1
332	—	0	0	0
530	—	0	0	0
103	—	0	1	0
422	—	0	1	0
113	—	0	0	0
441	—	0	0	0
600	—	0	0	0
610	—	0	0	0
531	—	0	0	0
213	—	0	0	0
432	1.446	15	9	4
620	—	6	3	0
611	1.430	15	2	0
512			7	3
540	—	0	1	0
223	—	0	1	0
303	—	0	0	0
313	1.388	5	5	1
621	1.379	22	6	3
522			13	9
541	1.365	7	5	2
630	—	0	0	0
323	—	0	0	0
442	—	0	0	0
631	—	0	1	1
532	1.306	50	47	42
550	1.287	55	19	17
710			1	0
413			51	54
602	1.281	23	24	24
333	1.275	30	27	30
612	1.268	17	17	21
640	—	0	1	0
701	—	0	3	3
720	1.253	40	55	47
423	—	0	0	1
551	1.245	17	18	25
711			0	0
622	1.233	10	11	15
641	1.222	10	2	3
542			8	14
721	1.211	20	26	40
730	—	0	0	0
433	—	0	1	5
503	—	0	1	0
004	1.187	20	54	56
513			1	6
632	—	0	3	4
114	—	0	0	0
650	—	0	6	6
731	—	0	1	3
523	—	0	2	4
204	—	0	0	0

pct Mn and 36.3 atomic pct Mo. This gives 19.1 Mn atoms and 10.9 Mo atoms per unit cell. There may be a slight excess of one of the pure elements. However, the composition could be limited with certainty to the range $Mn_{18}Mo_{12}$ to $Mn_{20}Mo_{10}$.

A powder photograph was taken with a rod-mounted specimen in a cylindrical camera with 5 cm radius, using Cr Kα. Intensities were estimated by visual comparison with a standard film strip. These data were not sufficiently accurate to warrant the application of absorption corrections. A small number of the intensities were also determined with a Geiger-counter spectrometer using Cu Kα and a thick flat sample, for which the absorption correction is constant for all angles. It was not possible to study very many reflections with the spectrometer because of the low intensities and high background encountered.

Table II. Geiger-Counter Data for Mn-Mo σ Phase (Cu Kα)

hkl	$I_{obs.}$	Ordered $I_{calc.}$	Disordered $I_{calc.}$
311	20	27	11
002	24	32	38
330	62	63	62
202	64	66	59
212	127	123	115
411	170	166	170
331	84	79	84
222	17	15	19
312	17	13	19

All calculated intensities have Lorentz, polarization, and multiplicity factors applied. Also, the atomic-structure factors used were corrected for anomalous dispersion.

Several models for ordering of the Mn and Mo atoms were tried. It was found that good agreement between calculated and observed intensities can be obtained only if I and IV contain all Mn atoms and II all Mo atoms. The ordered model which gave the best agreement for composition $Mn_{18}Mo_{12}$ is the following: 2 Mn in I; 4 Mo in II; 4 Mn, 4 Mo in III; 8 Mn in IV; and 4 Mn, 4 Mo in V. All data presented here are for $Mn_{18}MO_{12}$. However, good agreement can also be obtained for $Mn_{20}Mo_{10}$ by replacing two Mo atoms in position III by Mn atoms. If the composition is indeed $Mn_{19}Mo_{11}$, the only change indicated is in the ratio of Mn to Mo atoms in position III.

Observed and calculated intensities for both the disordered and ordered models are given in Table I (photographic data) and Table II (counter data). In general, the agreement is good for both models, but there are a few critical reflections, for which the ordered model is definitely better. Reflections (311) and (002) are observed as very nearly equal in intensity, in good agreement with calculated values for the ordered model, while the disordered model would make (002) more than three times as strong as (311). (202) is observed to be stronger than (330), a reversal of the disordered-model calculation; this is remedied in the ordered model. Reflections (432), (611) and (512), (313), and (541) would not be observed if the disordered model is correct; they are observed, however, and with intensities in fair agreement with those calculated for the ordered model.

This ordered model is also a reasonable one geometrically. The space available to an atom is large for positions II, which contain only Mo atoms, small for positions I and IV which contain only Mn atoms, and intermediate for positions III and V which accommodate both Mo and Mn atoms.

It is concluded from the X-ray studies cited, together with the geometrical consideration, that the σ phase in the Mn-Mo system is ordered in the manner designated above.

[1] Unpublished.

[2] B. F. Decker, R. M. Waterstrat, and J. S. Kasper: *Trans.* AIME (1953) 197, p. 1476; JOURNAL OF METALS (November 1953).

Developments in the Carbonate Processing of Uranium Ores

by F. A. Forward and J. Halpern

A new process for extracting uranium from ores with carbonate solutions is described. Leaching is carried out under oxygen pressure to ensure that all the uranium is converted to the soluble hexavalent state. By this method, alkaline leaching can be used successfully to treat a greater variety of ores, including pitchblende ores, than has been possible in the past. The advantages of carbonate leaching over conventional acid leaching processes are enhanced further by a new method which has been developed for recovering uranium from basic leach solutions. This is achieved by reducing the uranium to the tetravalent state with hydrogen in the presence of a suitable catalyst. A high grade uranium oxide product is precipitated directly from the leach solutions. Vanadium oxide also can be precipitated by this method. The chemistry of the leaching and precipitation reactions are discussed, and laboratory results are presented which illustrate the applicability of the process and describe the variables affecting leaching and precipitation rates, recoveries, and reagent consumption.

THE extractive metallurgy of uranium is influenced by a number of special considerations which generally do not arise in connection with the treatment of the more common base metal ores. Perhaps foremost among these is the very low uranium content of most of the ores which are encountered today, usually only a few tenths of one percent. A further difficulty is presented by the fact that the uranium often occurs in such a form that it cannot be concentrated efficiently by gravity or flotation methods. In these and other important respects, there is evident some degree of parallelism between the extractive metallurgy of uranium and that of gold and, as in the latter case, it has generally been found that uranium ores can best be treated directly by selective leaching methods.

It is readily evident that this parallel does not extend to the chemical properties of the two metals. Unlike gold, which is easily reduced to metallic form, uranium is highly reactive. It tends to occur as oxides, silicates, or salts. Two ores are of predominant importance as commercial sources of this metal: pitchblende which contains uranium as the oxide, U_3O_8, and carnotite in which the uranium is present as a complex salt with vanadium, $K_2O \cdot 2UO_3 \cdot V_2O_5 \cdot 3H_2O$. These ores may vary widely in respect to the nature of their gangue constituents. Some are largely siliceous in composition, while others consist mainly of calcite. Sometimes substantial amounts of pyrite or of organic materials are present and these may lead to specific problems in treating the ore. Further complications may be introduced by the presence of other metal values such as gold, copper, cobalt, or vanadium whose recovery has to be considered along with that of the uranium, or whose separation from uranium presents particular difficulty.

In general, there are two main processes for recovering uranium in common use today.[1,2] One of these employs an acid solution such as dilute sulphuric acid to extract the uranium from the ore. A suitable oxidizing agent such as MnO_2 or $NaNO_3$ is sometimes added if the uranium in the ore is in a partially reduced state. The uranium dissolves as a uranyl sulphate salt and can be precipitated subsequently by neutralization or other suitable treatment of the solution.

The second process employs an alkaline leaching solution, usually containing sodium carbonate. The uranium, which must be in the hexavalent state, is dissolved as a complex uranyl tricarbonate salt, and then is precipitated either by neutralizing the solution with acid or by adding an excess of sodium hydroxide. The latter method has the advantage of permitting the solutions to be recycled, since the carbonate is not destroyed. This is essential if the process is to be economical, particularly with low grade ores.

With each of these processes, there are associated a number of advantages and disadvantages and the choice between using acid or carbonate leaching is generally determined by the nature of the ore to be treated.

In the past, more ores appear to have been amenable to acid leaching than to carbonate leaching and the former process correspondingly has found wider application. With most ores, acid leaching has been found to operate fairly efficiently and to yield high recoveries. One of the main disadvantages has been that large amounts of impurities, such as iron and aluminum, sometimes are taken into solution along with the uranium. This may give rise to a high reagent consumption and to difficulties in separating a pure uranium product. Excessive reagent consumption in the acid leach process also may result

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from the presence in the ore of basic constituents such as calcium carbonate. Where the amount of such constituents is large, acid treatment is often uneconomical and carbonate leaching can be employed more profitably.

Among the chief advantages of the carbonate leach treatment are the relatively noncorrosive properties of the solution and its fairly specific solvent power for uranium and vanadium. Few impurities are dissolved and simple procedures are available for recovering high grade uranium and vanadium products from the leach liquors. The solutions usually can be recycled and the reagent consumption kept fairly low.

The main deterrent to the more extensive use of carbonate leaching in the past appears to have been the fact that it frequently yielded lower recoveries than could be obtained by acid leaching. With some ores, at least, it would appear that this has not been due to the limitations of the carbonate leaching method, but rather to deficiencies in the techniques which have been used in applying it.

It is the object of this paper to draw attention to some of the problems associated with the carbonate leaching process and to show how it has been possible, by introducing suitable pressure techniques, to extend greatly its efficiency and applicability to the point where it can be used with advantage to treat a greater variety of uranium ores. The usefulness of the process is enhanced further by the development of an entirely new method, also involving pressure techniques, which permits the recovery of high grade uranium and vanadium oxide precipitates directly from carbonate leach solutions.

In essence, these new developments are based on the consideration that compounds of hexavalent uranium are readily soluble in carbonate solutions, while those of lower valence are completely insoluble, and that the oxidation or reduction of uranium in carbonate solutions can be effected with gaseous molecular oxygen or hydrogen, respectively.

To ensure high uranium extractions, leaching is carried out at elevated temperatures under a positive partial pressure of oxygen or air, all the uranium thus being converted to and maintained in the soluble hexavalent state. By this means it has been found possible, for the first time, to apply carbonate leaching successfully to primary ores, such as pitchblende which contains uranium largely in reduced form and which formerly could be treated only with acid. This need for oxidizing the uranium appears to have been overlooked in most of the earlier carbonate leaching procedures with the result that poor recoveries were often obtained. The procedure described here, in which oxygen or compressed air is used to effect this oxidation, was suggested by earlier work which led to the development of a similar process for leaching Ni-Cu-Co sulphide ores with ammonia.[3]

Some of the considerations already referred to also suggested that reduction of uranium from the hexavalent to the tetravalent state would provide a particularly effective method of precipitating it from carbonate solutions. It was found that this reduction could be achieved best by treating the solution with gaseous hydrogen at elevated temperatures and pressures in the presence of a suitable catalyst. The reduced uranium is precipitated completely as the oxide, UO_2. This method of precipitation also applies to vanadium and appears to have certain advantages over the caustic precipitation procedure which is also considered in this paper.

The pressure leaching process and the caustic and hydrogen precipitation processes have been subjected to extensive laboratory studies with a view of establishing their applicability, examining their chemistry, and obtaining quantitative information about the variables which determine the leaching and precipitation rates, recoveries, and reagent consumptions. Some of the results of these investigations are presented and discussed in the following sections.

Experimental

Materials: Table I describes some typical ores on which leaching tests were conducted. They range in uranium content from 0.12 to 0.71 pct U_3O_8 and were selected to represent as wide as possible a range of grades and compositions. Most of the ores are of primary origin containing uranium principally as pitchblende or uraninite. A few secondary ores such as carnotite, are also represented (*B* and *D*).

The catalyst used in the hydrogen precipitation experiments was Eimer and Amend chemically pure nickel powder. Its surface area, determined by the BET adsorption method, was 0.72 sq m per g.

Commercial oxygen and hydrogen gases, supplied in cylinders, were used without purification. All other reagents were of chemically pure grade.

Procedures: *Leaching Experiments:* 1500 g of ore, crushed to −10 mesh, was rod-milled to the required particle size and made up with water and sodium carbonate to a desired pulp density and solution composition. The pulp was placed in an autoclave and maintained with continuous stirring at a given temperature and oxygen pressure. Samples of the ore and solution were withdrawn periodically and analyzed to determine the leaching rate, U_3O_8 extraction, and reagent consumption.

Caustic Precipitation Experiments: The solution was analyzed for U_3O_8, V_2O_5, Na_2CO_3, and $NaHCO_3$. Sufficient NaOH was added to neutralize the $NaHCO_3$, allow for the formation of sodium uranate, and provide a given excess in the solution. The solution was stirred and analyzed periodically to determine the extent of U_3O_8 precipitation.

Hydrogen Precipitation Experiments: The charge, consisting of 3 to 4 liters of solution of given composition and a desired quantity of nickel catalyst, was placed in an autoclave and maintained with continuous stirring at a given temperature and hydrogen pressure. The solution was analyzed periodically to determine the rate of uranium precipitation and the reagent consumption.

Analytical Procedures: Uranium was determined by the column method[4] or fluorimetrically.[5] Vanadium was determined by titration with $FeSO_4$.[6] $CO_3^=$, HCO_3^-, and OH^- were determined by titration with standard HCl.[7]

Results and Discussion

Carbonate Leaching: *Leaching Rates and U_3O_8 Recoveries:* Minerals such as carnotite, which contains uranium in the hexavalent form, generally are dis-

Table I. Analysis of Typical Ores

Ore	*A*, Pct	*B*, Pct	*C*, Pct	*D*, Pct	*E*, Pct	*F*, Pct
U_3O_8	0.12	0.16	0.18	0.31	0.41	0.71
V_2O_5	nil	0.15	nil	0.91	nil	nil
S	0.25	nil	0.50	nil	0.50	0.30
$CaCO_3$	11.36	77.5	11.36	8.5	13.64	26.8
Fe	2.0	trace	3.8	1.6	3.9	6.9
Acid insoluble	88.0	18.0	88.0	86.8	86.5	60.4

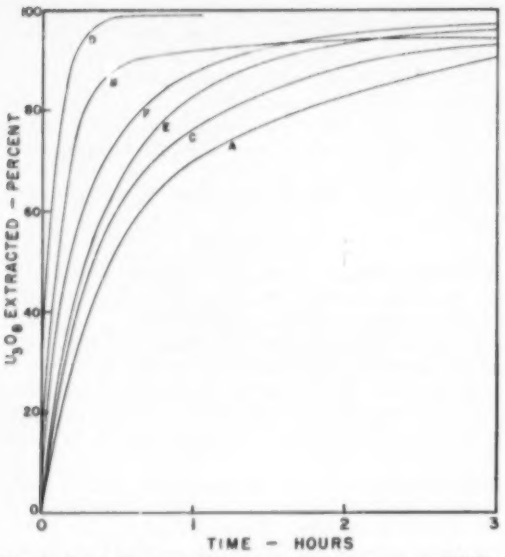

Fig. 1—Leaching rates of various uranium ores at 100°C, 30 psi O_2.

solved readily by solutions containing carbonate and bicarbonate salts according to the following equation

$$K_2O \cdot 2UO_3 \cdot V_2O_5 \cdot 3H_2O + 2CO_3^{=} + 4HCO_3^{-} \rightarrow 2K^{+} + 2UO_2(CO_3)_3^{\equiv} + 2VO_3^{-} + 5H_2O. \quad [1]$$

With primary minerals such as pitchblende, similar dissolution should occur providing oxygen is present to convert the uranium to the soluble hexavalent form. The leaching reaction then may be written as

$$U_3O_8 + \tfrac{1}{2}O_2 + 3CO_3^{=} + 6HCO_3^{-} \rightarrow 3UO_2(CO_3)_3^{\equiv} + 3H_2O. \quad [2]$$

Some bicarbonate as well as carbonate must be present in the solution to prevent the formation of free OH^- during leaching. If the solution contains carbonate only, the accumulation of OH^- results in the hydrolysis of the uranyl ion and its reprecipitation as insoluble uranate salts.

In accordance with these considerations, it was found that, in the presence of oxygen, high uranium extractions were obtained by carbonate-bicarbonate leaching on all the ores examined. Some typical leaching rate curves are shown in Fig. 1. The final U_3O_8 extractions for the various ores and the corresponding retention times are summarized in Table II.

Table II. Summary of Retention Times and Final U_3O_8 Extractions*

Ore	U_3O_8 Analysis, Initial, Pct	U_3O_8 Analysis, Final, Pct	Retention Time, Hr	Final U_3O_8 Extraction, Pct
A	0.12	0.010	4	92.0
B	0.16	0.008	3	95.0
C	0.18	0.013	3	93.0
D	0.31	0.003	1	99.0
E	0.41	0.012	3	97.0
F	0.71	0.032	3	95.5

* Average mesh of grind, 90 pct −325 mesh; solution, Na_2CO_3: 40 to 50 g per liter, $NaHCO_3$: 10 to 20 g per liter; temperature, 100°C; O_2 partial pressure, 2 atm; and pulp density, 25 to 60 pct solids.

Table III. Reagent Consumption During Carbonate Leaching of Various Ores*

Ore	Sulphur Content, Initial, Pct	Sulphur Content, Final, Pct	Na_2CO_3 Consumed,† Lb per Ton	Na_2SO_4 Equivalent, Lb per Ton	Na_2SO_4 Produced, Lb per Ton
A	0.25	0.10	20.4	13.7	12.8
B	nil	nil	nil	nil	nil
C	0.50	0.20	38.2	25.6	26.8
D	nil	nil	nil	nil	nil
E	0.50	0.20	37.9	25.4	26.7
F	0.30	0.06	31.8	21.3	n.d.
E‡	0.18	0.06	13.2	8.8	8.5

* Same conditions as in Table II.
† Converted to $NaHCO_3$, chiefly by Eq. 3.
‡ After partial removal of sulphides by flotation.

It is seen that, while all the ores yielded final U_3O_8 extractions in excess of 92 pct, the retention times required to obtain these recoveries, under fixed leaching conditions, varied from 1 to 4 hr for the different ores. In general, leaching rates were higher with the secondary ores where only a physical dissolution process was involved than with the primary ores where the chemical oxidation process was rate controlling.[8]

Reagent Consumption in Side Reactions: The amounts of carbonate, bicarbonate, and oxygen consumed during leaching, by reaction with the uranium as shown by Eqs. 1 and 2, usually are very small. However, it was found that side reactions with other constituents of the ores often gave rise to additional consumption of these reagents which, in some cases, was more substantial. Some of the reactions, which were encountered, are illustrated by the following equations

$$2FeS_2 + 7\tfrac{1}{2}O_2 + 8CO_3^{=} + 7H_2O \rightarrow 2Fe(OH)_3 + 4SO_4^{=} + 8HCO_3^{-}. \quad [3]$$

$$CaSO_4 + Na_2CO_3 \rightarrow CaCO_3 + Na_2SO_4. \quad [4]$$

$$SiO_2 + H_2O + 2CO_3^{=} \rightarrow SiO_3^{=} + 2HCO_3^{-}. \quad [5]$$

$$CaO + 2HCO_3^{-} \rightarrow CaCO_3 + H_2O + CO_3^{=}. \quad [6]$$

Table III shows that, with most of the ores that were examined, the observed reagent consumption was due largely to the reaction of sulphides in accordance with Eq. 3. Where the extent of this reaction was excessive, it was found that the sulphide content of the ore, and consequently the reagent consumption on leaching, usually could be reduced to a moderate value without appreciable loss of uranium by prior flotation. Dissolution of silica (Eq. 5) was negligible at temperatures below 120°C.

Carbonaceous matter, which was present in some of the ores, also reacted with oxygen and carbonate during the leach, consuming these reagents and contaminating the leach solutions. This usually could be avoided by subjecting the ore to a low temperature roast (400° to 600°C) prior to leaching. Often an improvement in the settling and filtering characteristics of the ore also resulted from this roast treatment.

Leaching Variables: The leaching rates and recoveries of uranium are influenced by various conditions such as the grind of the ore, the carbonate and bicarbonate concentrations of the leach solution, temperature, and oxygen pressure. A detailed dis-

cussion of these variables has been presented in earlier papers.[8, 9] Some of the more important effects are summarized here:

1—Grind: With pitchblende ores, grinding to −100 mesh was generally necessary to obtain maximum recoveries. The leaching rate increased with the extent of grinding so as to suggest that it was directly proportional to the exposed surface area of the pitchblende.[8, 9] Carnotite ores could be leached at a somewhat coarser grind, usually corresponding to the natural grain size of the ore.

2—Composition of the Leach Solutions: Optimum leaching rates and recoveries were generally obtained when the solution contained 40 to 50 g per liter Na_2CO_3 and 10 to 20 g per liter $NaHCO_3$. At lower concentrations, the leaching rate fell off, while higher concentrations appeared to offer no particular advantage. It was found that the presence of some bicarbonate in the solutions was particularly critical for efficient leaching. Neutral salts such as Na_2SO_4, which were sometimes formed during leaching and accumulated in the leach solutions as they were recycled, were found to be without effect.

3—Pulp Density: Varying the pulp density from 25 to 60 pct solids was found to have no effect on the leaching rates and recoveries of uranium from any of the ores.

4—Temperature: The leaching rates for the pitchblende ores were found to have a fairly high temperature coefficient, increasing by 50 to 100 pct for every rise in temperature of 10°C. This is illustrated in Fig. 2 which depicts the rate curves for a typical ore at various temperatures ranging from 61° to 115°C. It is seen that leaching becomes very slow below 80° to 100°C and that for optimum results the temperature should be maintained at about 100°C. Excessively high temperatures (above 120°C) should be avoided, since they give rise to the dissolution of appreciable amounts of silica. The high temperature coefficient of the leaching rate for pitchblende ores reflects the fact that the rate is determined by a chemical reaction, involving the oxidation of the uranium. For some of the carnotite ores in which the uranium occurs in a fully oxidized form and only a physical dissolution process appears to be involved, the temperature coefficient of the leaching rate was found to be lower and correspondingly lower leaching temperatures of 60° to 70°C could be employed.

5—Oxygen: With all the pitchblende ores, the presence of oxygen was found to be critical to the leaching process. Some results illustrating this are shown in Fig. 3. It is seen that no leaching occurred in the absence of oxygen. As the oxygen pressure was increased from 0.5 to 4.0 atm, the leaching rate

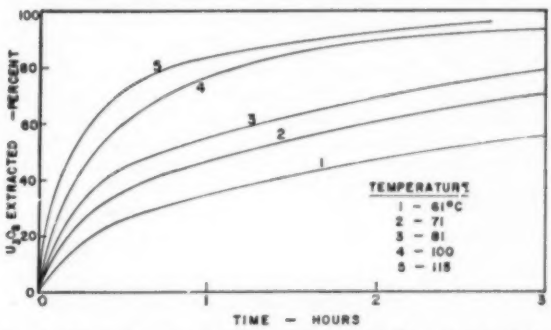

Fig. 2—Effect of temperature on carbonate leaching of a typical pitchblende ore under oxygen pressure.

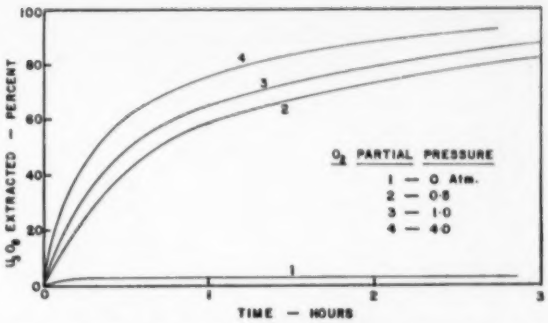

Fig. 3—Effect of O_2 pressure on carbonate leaching of a pitchblende ore at 100°C.

increased in direct proportion to the square root of the pressure. Results such as these serve to confirm the fact that oxidation of the uranium is necessary before it becomes soluble in the carbonate leach solutions, and, further, that this oxidation process tends to determine the leaching rate.[9] At 100°C, an oxygen partial pressure of about 1 atm (75 psi air) appeared to give rapid oxidation and leaching. At lower oxygen pressures, correspondingly longer retention times are required.

With some carnotite ores, high uranium recoveries could be obtained by carbonate leaching even in the absence of oxygen. With others, leaching under oxygen pressure resulted in increased extractions, indicating that at least part of the uranium was present as pitchblende or other primary minerals or that the ore contained reducing compounds such as sulphides or carbonaceous matter which reduced and insolubilized some of the uranium. It is clear that, in any operation involving carbonate leaching of uranium ores, particular consideration should be given to ensuring adequate oxidizing conditions.

In achieving these conditions, the procedure described here involving the use of oxygen under pressure would appear to be favored by a number of factors. Among these are the relatively low cost of oxygen and its effectiveness as an oxidizing agent for uranium minerals and compounds in carbonate solutions, demonstrated by extensive laboratory results.[8, 9] The oxygen can be supplied as compressed air and in this form would appear to be considerably cheaper than chemical oxidizing agents of comparable efficiency.

In this connection, it might be noted that most of the common oxidizing agents such as sodium chlorate, sodium nitrate, and manganese dioxide which are used to oxidize uranium in acid media are not effective in alkaline solutions. More expensive reagents such as potassium permanganate must be employed and the cost entailed in their use may become a serious consideration, particularly since substantial amounts are often consumed in side reactions with other reducing constituents such as sulphides contained in the ore.

The use of oxygen is further favored by the fact that the only product of its reaction is water. The introduction of contaminants into the leach solutions, which might result from the use of chemical oxidizing agents, thus is avoided. This may be of importance where the solutions are to be recycled.

Recovery of Vanadium: In treating carnotite ores, it is often necessary to make provision also for the economic recovery of vanadium values. The problem of extracting vanadium by carbonate leaching has recently been discussed by Butler,[10] who reported

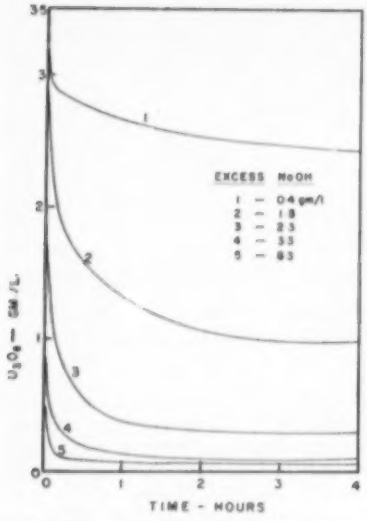

Fig. 4—Effect of NaOH on precipitation of uranium from carbonate solutions. Initial U_3O_8, 3.5 g per liter. Na_2CO_3, 50 g per liter.

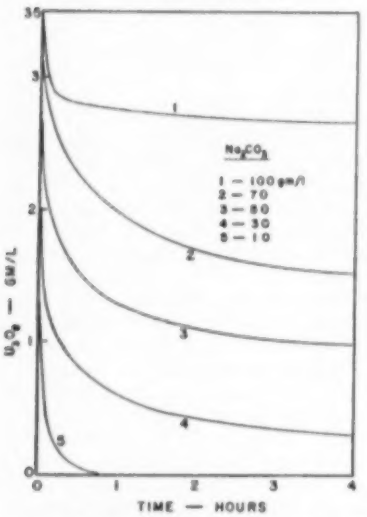

Fig. 5—Effect of Na_2CO_3 concentration on caustic precipitation of uranium. Initial U_3O_8, 3.5 g per liter. Excess NaOH, 1.3 g per liter.

recoveries of up to 85 pct with certain ores, following suitable roast treatment. In general, carbonate leach solutions from the treatment of carnotite ores will contain both uranium and vanadium values. The methods which may be used to separate these values from the solutions will be considered.

Precipitation of Uranium and Vanadium from Carbonate Solutions: *Neutralization:* The following description is illustrative of most of the practical methods which have been used in the past to recover both uranium and vanadium from carbonate leach solutions:[10-13]

The solution generally is acidified to a pH of about 6 and boiled to expel the CO_2. All the uranium and a part of the vanadium thereby are precipitated as a yellow cake approximating to sodium uranyl vanadate in composition (45 pct U_3O_8, 17 pct V_2O_5 dry basis). This product is fused subsequently with a mixture of soda ash, salt, and sawdust and the fusion cake leached with water. This dissolves the vanadium and yields a high grade uranium oxide product as the residue. The filtrates from the uranium precipitation and from the uranium refining operations are further acidified to a pH of about 2.5 and heated to precipitate a vanadium oxide red cake. The final vanadium oxide product is obtained by fusing this precipitate.

While reasonable recoveries and products can be obtained by this procedure, it is often unsatisfactory because of the large number of carefully controlled steps which are involved and because the neutralization of the solutions and destruction of the carbonate precludes recycling of the barren liquors. Substantial amounts of sodium carbonate and acid thus are consumed in the treatment of the ore.

Caustic Precipitation: Where the carbonate solution contains only a minor amount of vanadium which does not have to be recovered, precipitation of the uranium may be achieved without neutralizing the carbonate by raising the pH of the solution to a value of 12 or higher through the addition of sodium hydroxide.[3] Sufficient sodium hydroxide is required to react with the bicarbonate in the solution, to allow for the hydrolysis of the uranium, and to provide an excess of 5 to 10 g per liter free NaOH. The uranium then is precipitated as a mixture of sodium uranates according to the following equations

$$UO_2(CO_3)_3^{=} + 2Na^+ + 4OH^- \rightleftarrows Na_2UO_4 + 3CO_3^{=} + 2H_2O. \quad [7]$$

$$2UO_2(CO_3)_3^{=} + 2Na^+ + 6OH^- \rightleftarrows Na_2U_2O_7 + 6CO_3^{=} + 3H_2O. \quad [8]$$

It is likely that some polyuranates of more complex composition are formed also. The dried precipitate generally contains about 70 pct U_3O_8.

As would be suggested by the equilibria in Eqs. 7 and 8, complete precipitation is favored by a high concentration of free OH^- and Na^+ ions in the solution and prevented by a high concentration of $CO_3^=$. Some results of laboratory experiments which confirm this are given in Figs. 4, 5, and 6.

It is seen that, with a solution containing 50 g per liter of Na_2CO_3 and 3.5 g per liter of U_3O_8, addition of an excess of about 3 g per liter of NaOH precipitates the uranium quantitatively (>97 pct) in 4 hr. With increasing Na_2CO_3 concentration, the efficiency diminishes and correspondingly larger amounts of NaOH must be provided to achieve the same degree of precipitation, so that it is doubtful whether the method would be effective with solutions containing more than 80 to 100 g per liter of Na_2CO_3. On the other hand, neutral sodium salts such as Na_2SO_4 favor precipitation. Other factors contributing to efficient precipitation are a high initial U_3O_8 concentration, seeding of the solutions, and agitation. Rais-

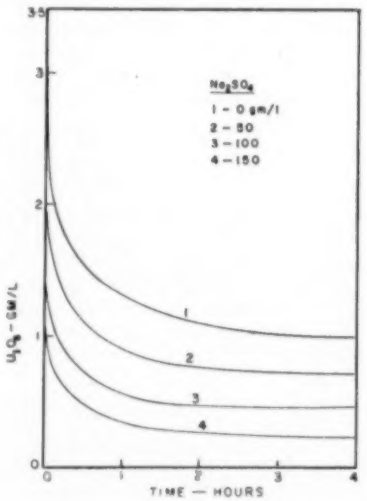

Fig. 6—Effect of sodium salt concentration on caustic precipitation of uranium. Initial U_3O_8, 3.5 g per liter, Na_2CO_3, 50 g per liter. Excess NaOH, 1.3 g per liter.

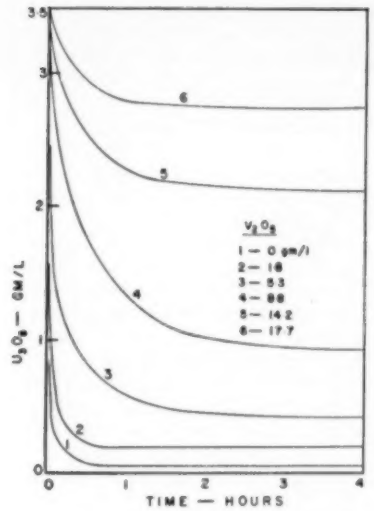

Fig. 7—Effect of vanadium on caustic precipitation of uranium. Initial U_3O_8, 3.5 g per liter. Na_2CO_3, 50 g per liter. Excess NaOH, 5 g per liter.

ing the temperature of the solution was found to have a markedly detrimental effect.

The chief advantages of the caustic precipitation method lie in its relative simplicity of operation and in the fact that the carbonate is not destroyed. The solutions therefore can be recycled, provided that the excess NaOH is first neutralized and the $NaHCO_3$ required for leaching is supplied by adding CO_2, $NaHCO_3$, or H_2SO_4 to the barren liquor. Where the U_3O_8 content of the pregnant liquor is fairly high (>2 g per liter) and the concentrations of Na_2CO_3 and $NaHCO_3$ not excessive, precipitation by this method is rapid and complete and the reagent consumption moderate.

Vanadium, if present in the solution, is not recovered by the caustic precipitation method, except for minor amounts which may coprecipitate with the uranium. However, it was found that in high concentrations vanadium interferes with the precipitation of uranium to such an extent that the method becomes impractical.

Some results illustrating this are given in Fig. 7. It is seen that the same conditions, which were effective in precipitating uranium quantitatively in the absence of vanadium, gave practically no precipitation from a solution containing 17.7 g per liter of V_2O_5. Even when the concentration of free NaOH was increased from 4 to 20 g per liter, about 1 g per liter of U_3O_8 remained unprecipitated. Thus, the method does not appear to be suitable for precipitating uranium from solutions containing more than 2 to 3 g per liter of V_2O_5, even when provision does not have to be made for recovering the vanadium.

Precipitation by Hydrogen Reduction: Such objections and limitations, as are associated with the two precipitation procedures described above, are overcome for the most part in the hydrogen precipitation method.[13, 14] In this procedure, the carbonate solution is heated under hydrogen pressure to reduce the uranium and vanadium salts to insoluble lower valence states. The oxides of both metals are precipitated according to the following equations

$$UO_2(CO_3)_3^{=} + H_2 \rightarrow UO_2 + CO_3^{=} + 2HCO_3^{-}. \quad [9]$$

$$2VO_3^{-} + 2H_2 + 2HCO_3^{-} \rightarrow V_2O_3 + 2CO_3^{=} + 3H_2O. \quad [10]$$

A suitable hydrogenation catalyst such as metallic nickel powder is provided to ensure rapid reaction and complete precipitation at moderate temperatures and pressures.

The course of a typical precipitation experiment employing this procedure is depicted in Fig. 8. A solution containing initially 2.5 g per liter U_3O_8, 7.5 g per liter V_2O_5, 50 g per liter Na_2CO_3, and 20 g per liter $NaHCO_3$ was treated in an autoclave at 150°C with gaseous hydrogen maintained at a partial pressure of 200 psi. 5 g per liter nickel powder was used as catalyst. It is seen that both the uranium and vanadium were precipitated in about 4½ hr, the final concentrations being 0.04 g per liter U_3O_8 and 0.06 g per liter V_2O_5.

In this case, the product obtained directly from the autoclave comprised a mixture of uranium and vanadium oxides and the unchanged metallic nickel powder. The nickel catalyst was removed readily with a magnetic separator and recycled. The uranium and vanadium oxides in the nickel-free product were separated by the fusion procedure described earlier.

With leach solutions containing uranium only, a high grade uranium oxide product can be obtained directly from the autoclave and the need for the subsequent fusion and separation treatment is eliminated.

The kinetics of the hydrogen precipitation reactions and the variables which influence the proc-

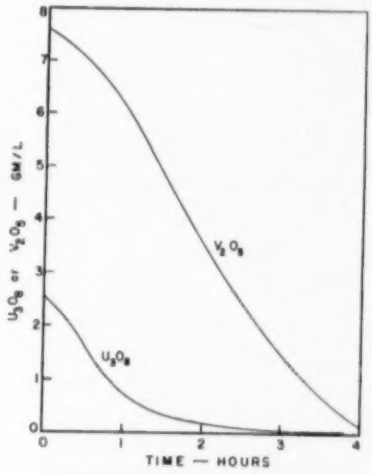

Fig. 8—Precipitation of uranium and vanadium from a carbonate solution by hydrogen reduction. Temperature, 150°C; H_2 pressure, 200 psi. Ni, 5 g per liter.

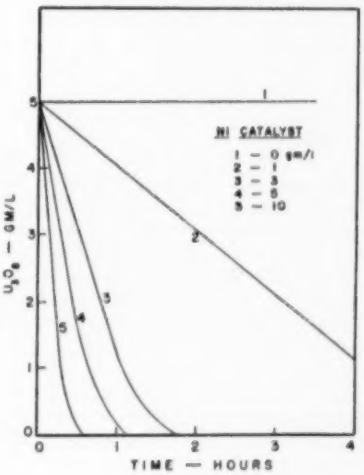

Fig. 9—Effect of nickel catalyst on precipitation of uranium by hydrogen reduction. Temperature, 106°C; H_2 pressure, 500 psi.

ess have been discussed in detail in earlier papers.[13, 14] It was shown that the rates of precipitation of both uranium and vanadium increase with temperature and hydrogen pressure and are directly proportional to the amount of catalyst present (see Fig. 9). For practical operation, the precipitation must be carried out in a pressure vessel at temperatures of 100° to 150°C and hydrogen partial pressures of 50 to 200 psi. Varying the amount of catalyst provides a particularly convenient and inexpensive method of controlling the rate of precipitation, since the catalyst itself is not consumed and may be recycled indefinitely.

One of the most advantageous features of the hydrogen precipitation process is its general applicability. In combination with the fusion procedure described earlier, it can be used to recover uranium and vanadium as high grade products and in high yields from nearly any carbonate leach solution. The carbonate and bicarbonate concentrations and the initial concentrations of uranium and vanadium are not critical, and most of the impurities which are likely to be present in such solutions do not interfere.

The reagent consumption in hydrogen precipitation is generally very small, being equivalent only to the amount of uranium and vanadium. The carbonate and bicarbonate concentrations of the solutions are not altered appreciably during precipitation and no impurities are introduced. The barren solutions thus are ideally suited for recycling in further leaching and precipitation stages. This constitutes an advantage in the use of hydrogen over that of other chemical reducing agents or precipitation reagents. The facts that complete precipitation is achieved and that the precipitation time as well as the reagent consumption are proportional to the amounts of uranium and vanadium precipitated make the method particularly attractive where low grade leach solutions are involved.

The chief disadvantage of the process, in relation to the other precipitation procedures, lies in the fact that it must be operated at higher temperatures and pressures. This is offset to some extent by the short retention times which can be achieved and the corresponding reduction in the volume of the reaction vessels. The heat requirements can be kept low by the use of suitable heat exchangers.

Conclusions

An attempt has been made in this paper to examine the problems involved in the carbonate leach treatment of uranium ores and to describe new pressure leaching and precipitation techniques which show promise of improving the efficiency and extending the applicability of this method of treatment.

The lower uranium recoveries, which carbonate leaching has often provided in the past in comparison with acid leaching, have been attributed to failure to make provision for the oxidation of the uranium to the soluble state. An effective procedure for achieving this oxidation has been described in which leaching is carried out under oxygen or air pressure. The chemistry of the leaching reactions has been discussed and results have been presented which show that this procedure yields consistently high uranium extractions from a variety of ores, including primary uranium ores such as pitchblende, which were not previously considered to be amenable to carbonate leaching.

Another disadvantageous feature which sometimes has been associated with the carbonate leach treatment has been the large reagent consumption and difficulty involved in precipitating the uranium and vanadium from the leach solutions. Frequently the solutions had to be neutralized and adequate recycling was impossible. A new precipitation procedure has been described here which overcomes these difficulties and further contributes to the usefulness of the carbonate leach treatment. Uranium and vanadium oxides are precipitated quantitatively from the leach solution by treating the solution with hydrogen under pressure in the presence of a nickel catalyst. The method is applicable to nearly any carbonate leach solution and can be used to recover either uranium or vanadium alone or both metals from the same solution. Yields are quantitative and the products are of particularly high grade. The reagent consumption is very low and the solutions can be recycled.

With improved techniques such as these, it is suggested that carbonate leaching, which is characterized by many inherently advantageous features, should find useful application in the treatment of many uranium ores.

Acknowledgments

Grateful acknowledgment is made by the authors to the National Research Council of Canada, Atomic Energy Control Board, and Eldorado Mining & Refining Co. for research grants which made possible the work described in this paper. Appreciation is also expressed for the technical assistance provided by the staff of the Mines Branch, Ottawa, Ont., Canada.

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aime NEWS

Final Drive to Complete Annual Meeting Plans Nears Culmination

Annual Meeting technical program planning is well on its way for the 1955 gathering at the Hotel Conrad Hilton, Chicago, February 14 to 17.

The Hydrometallurgy session of the Extractive Metallurgy Div. has scheduled *Agitation versus Process Results in a Zinc Dust Purification*, by L. A. Painter and R. K. Carpenter. The committee is awaiting final word on three other papers, one of which will deal with tank room acidity control in electrolytic zinc production.

Among the papers to be presented at the Physical Chemistry of Extractive Metallurgy session are: *Electrical Conduction in Molten Copper-Iron Sulphide Mattes*, by G. M. Pound, G. Derge, and G. Osuch; *Thermodynamics of the Cu-Fe-S System at Matte-Smelting Temperatures*, by W. A. Krivsky and R. Schuhmann, Jr.; *Oxygen Solubility in Liquid Cu-Fe-S-O Mattes*, by H. L. Lander and R. Schuhmann; and *Sulphur Pressure Measurements of Molybdenum Sesquisulfide in Equilibrium with Molybdenite*, by C. L. McCabe.

Joint sessions are being arranged on physical chemistry for the Iron and Steel Div. and Extractive Metallurgy Div., and on hydrometallurgy for the Extractive Metallurgy Div. and Minerals Beneficiation Div.

Several outstanding papers have been scheduled for the Titanium session. Included on the list of expected presentations are: *The ONR Program of Research on Extractive Metallurgy of Titanium*, by W. C. Arsem and J. J. Harwood; *Preparation and Chlorination of Titaniferrous Slags from Idaho Ilmenites*, by A. H. Roberson and L. H. Banning; and *A New Method for Casting Titanium*, by R. A. Beall, A. H. Roberson, and F. W. Wood.

Two papers have already been set for the Titanium-Zirconium session. They are: *The Preparation of High Purity Zirconium Tetrachloride from Alkali Chlorozirconates*, by R. V. Harrigan; and *Zirconium and Hafnium Production Operations of the Carborundum Metals Corp.*, by W. W. Stephens and C. Q. Morrison.

One of the sessions to be staged by the Iron and Steel Div. that is expected to attract unusual interest is the one on New Processes in the Steel Industry. Among the papers to be heard will be: *Oxygen in the Bessemer Converter*, by F. J. McMulkin; and *One Year of Experiment with the International Low Shaft Furnace*, by P. Coheur, et al. Highlight of the program will be the presentation of the Howe Memorial Lecture by J. S. Marsh. He will speak on *Technological Developments in the Steel Industry*.

The Chicago Section of the National Open Hearth Committee is sponsoring a session on Basic Furnaces and Refractories. Papers will include: *Design and Operation of All Basic Open Hearth Furnaces at Keystone Steel & Wire Co.*, by A. H. Sommer; *Operation and Performance of all Basic Open Hearth Furnaces at U. S. Steel Corp.*, South Works, by J. J. Oravec and J. F. O'Brien; and *Behavior and Testing of Basic Refractories for Open Hearth Use*, by W. S. Debenham.

The new Committee of the Iron and Steel Div., Mechanical Working, has a session planned built around the following papers: *Forgeability of Steels*, by Anderson, et al.; *Evaluation of Soaking Pit Capacity*, by Robinson and Eckholm; *Factors in Soaking Pit Heating*, by R. Hindson; and *Practical Observations on the Influence of Slab Heating and Rolling Techniques on Slab Furnace*, by H. B. Wishhart.

One paper to be presented at the Powder Metallurgy session will evaluate disassociated ammonia and hydrogen atmospheres for stainless steel sintering. Relation of strength, composition and grain sizes of sintered WC-Co alloys will also be on the program. Another paper is expected to cover microstructure of titanium carbide.

Some of the best talent from Ed Sullivan's Sunday night television show, the Olson and Johnson stag show, and other great entertainment attractions will be featured at the 1955 dinner-smoker. Preview reports say that the show will be one of the best ever presented at an Annual Meeting. Before the smoker gets underway, however, there will be a cocktail party for one and all in the South Exhibition Hall of the hotel. Registration identification is all that is needed for admission. The cocktail party is sponsored by a group of companies.

Tickets for the dinner-smoker are scaled at $10.00 each. The Metals Branch Dinner will take place on Tuesday, tickets will cost $6.00. The EMD luncheon is scheduled for Wednesday, $3.50. A Powder Metallurgy Committee luncheon will be held on Thursday, $3.50.

Around the Sections

MINNESOTA

Approximately 90 people gathered to hear President elect H. DeWitt Smith address the Minnesota School of Mines and Metallurgy Society and Minnesota Local Section of AIME, at a dinner meeting held on October 25. The topic of Mr. Smith's talk was *Mineral Industry of the World*.

Mr. Smith discussed his experiences as a mining engineer and an executive.

PHILADELPHIA

The Philadelphia Section AIME met on Oct. 21, 1954 for a dinner in which both the mining and metallurgical members participated. Following the dinner, each group attended separate lectures.

The metallurgical group held its Annual Student Paper Night at the University of Pennsylvania. Medals and certificates were awarded to the following students: first prize—Donald R. Haines, Drexel Institute of Technology, *The Isothermal Transformation of a Chromium-Molybdenum Cast Steel;* second prize—A. R. Henricks, Drexel Institute of Technology, *The Effect of Innoculant Temperature Upon the Structure of Gray Cast Iron.*

1955 Publications Policies Established

Pursuant to Article X of the bylaws of the AIME, the following information is hereby given as to the "conditions, prices, and terms under which the various classes of members, and Student Associates, severally, shall be privileged to obtain publications of the Institute during the ensuing year."

Publications authorized for issue in 1955 include the following: MINING ENGINEERING, published monthly, containing material, including technical papers, of interest to those engaged in exploration, mining geology and geophysics; metal, nonmetallic, and coal mining and beneficiation; and fuel technology. The JOURNAL OF METALS, published monthly, containing material, including technical papers, of interest to those engaged in nonferrous smelting and refining, iron and steel production, and physical metallurgy. The JOURNAL OF PETROLEUM TECHNOLOGY, published monthly in Dallas, containing material, including technical papers, of interest to those engaged in petroleum and natural gas drilling and production.

Annual subscriptions to any one of the above journals will be provided all members in good standing without further charge, a subscription credit of $6 for members and $4 for Student Associates being included in the dues paid. (A member ceases to be in good standing if current dues are not paid by April 1). If more than one of the monthly journals is requested, $4 extra will be charged for an annual subscription, or 75¢ for single copies of regular issues and $1.50 for special issues. The nonmember subscription price for each journal is $8 in the Americas and U. S. possessions; foreign, $10, and for single issues 75¢ in the Americas and $1.00 foreign for regular issues and $1.50 for special issues. Student Associates will be entitled to the same privileges for all publications as members. AIME members subscribing to more than one of each of the three monthly journals will be billed at the nonmember rate of $8 per year, domestic; $10 foreign, for the extra subscription(s).

Three volumes of *"Transactions"* are authorized for 1955 publication, as follows: No. 199, Mining Branch; No. 200, Metals Branch; and No. 201, Petroleum Branch. Volumes 199 and 201 will be available to members at $3.50 each for a first copy if paid for in advance with dues; otherwise at the nonmember rate of $7 less 30 pct. Nonmembers $7 in the United States; foreign $7.50. Volume 200 will be available to members at $4.50 each for a first copy if paid for in advance with dues; otherwise at the nonmember rate of $9 less 30 pct. Nonmembers $9 in the United States, foreign $9.75.

Special volumes now planned for publication in 1955 include the following: *Open Hearth Proceedings*, Vol. 38, price to AIME members $7; nonmembers, $10. *Blast Furnace, Coke Oven, and Raw Materials Proceedings*, Vol. 14, AIME members $7; nonmembers $10. *Electric Furnace Steel Proceedings*, Vol. 12, AIME members $7; nonmembers $10. *Statistics of Oil and Gas Development and Production*, Vol. 8, covering data for the year 1953, members $5, nonmembers $10. AIME *Directory and Yearbook*, or equivalent free to members.

If dues are paid subsequent to January 31, back issues of Institute publications will be supplied only if adequate stocks are on hand. A member is not entitled to receive a volume of *"Transactions"*, or a special volume, in lieu of a monthly journal, free of charge on membership. Members in arrears for dues are not entitled to special members' prices for publications.

Rocky Mountain Members may have their choice of an annual subscription to one of the monthly journals on request.

Dues Bills In Mail

Pursuant to Article II, Section 2, of the bylaws of the AIME, notice is hereby given that dues for the year 1955 are payable Jan. 1, 1955, as follows: Members and Associate Members, $20; Junior Members for the first six years of Junior Membership, $12, and thereafter, $17; Student Associates (including an annual subscription to a monthly journal), $4.50.

Dues bills were mailed during the latter part of November. Prompt payment will assure uninterrupted receipt of the publications desired in 1955. If, for any reason, a bill is not received within a reasonable time, headquarters should be notified.

Institute Names 1955 Legion of Honor

Each year at the Annual Banquet in February, those members of the AIME who have continuously maintained their membership for 50 years are given special recognition. They are seated at the head table as guests of the Institute and are added to the membership of the Legion of Honor. The names of those who will achieve this status in 1955 are as follows:

Agaberg, Edgar C.
Bengal, India
Austin, Albert M.
New York, N. Y.
Blomfield, A. L.
Kirkland Lake, Ont., Canada
Carpenter, Arthur Howe
Chicago, Ill.
Cowans, Frederick
Pasadena, Calif.
Dilworth, John B.
Upper Darby, Pa.
Fink, William N.
Chihuahua, Mexico
Foote, Arthur B.
Grass Valley, Calif.
Fox, Alfred, Jr.
Birkenhead, England
Garrey, George H.
Denver, Colo.
Hamilton S. Harbert
Philadelphia, Pa.
Hammer, William L.
Corpus Christi, Texas
Harris, Henry
Worthing, Sussex, England
Hawxhurst, Robert, Jr.
San Francisco, Calif.
Kelley, Cornelius F.
New York, N. Y.
Krumb, Henry
New York, N. Y.
Rakowsky, Victor
Joplin, Mo.
Speller, F. N.
Pittsburgh, Pa.
Vail, Richard H.
Panajachel, Guatemala
Vallat, Benjamin W.
Pacific Palisades, Calif.
Wang, C. Y.
New York, N. Y.
Williams, R. Y.
Pottsville, Pa.
Wright, Charles Will
Washington, D. C.
Wyer, Samuel S.
Columbus, Ohio

Membership Emblem Proposed by Institute

On November 1, a newly designed emblem was instituted by the AIME Membership Steering Committee. This button was proposed as a tangible recognition to individual members who are successful in securing five membership applications for the grades of Member, Associate or Junior Member.

Applications now carry a credit line on which the Member responsible for getting the application should insert his name. The Institute Activities Dept. will keep score and make the presentation on a suitable occasion.

The first button recognizes ability to gather in five applications. Further recognition for 10, 15, 20, and up, in multiples of five await those energetic enough to do the superlative.

General Electric Announces

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- 3 MODELS
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Check the list at right for the variety of mountings possible with the OX-175. Then call the G-E x-ray representative near you for complete information on this latest development in industrial x-ray. Or write X-Ray Department, General Electric Company, Milwaukee 1, Wis., for Pub. AY124.

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  With elevating tube stand
  Rotating tube stand
  Tube carrier for 360° unit
- **BRIDGE CRANE**
  15—30-foot, with pantograph and motor-operated hoist.
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  Automatic
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- **ELEVATING TUBE STAND**
  Mobile base
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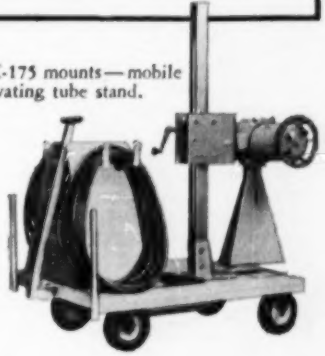
One of ten OX-175 mounts — mobile truck with elevating tube stand.



Progress Is Our Most Important Product

GENERAL ELECTRIC

Nominating Committee For AIME Officers, 1956

The following have been named by the Council of Section Delegates, the Branch Councils, and the President of the Institute to constitute the Nominating Committee for AIME Officers in 1956. The Committee will meet during the Annual Meeting of the Institute in Chicago, Feb. 14 to 17, 1955 and select the official slate. If the principal finds it impossible to attend, the alternate will act in his place; otherwise the alternate will not be present at the meeting of the Committee. The names of the alternates are given in parentheses.

President Reinartz's Appointments: Edward G. Fox, Chairman, Philadelphia & Reading Coal & Iron Co., Philadelphia, Pa. (Frank Ayer, New York, N. Y.); Charles E. Golson, Colorado Fuel & Iron Corp., Denver, Colo. (Wayne Dowdey, Eimco Corp., Birmingham, Ala.); Richard M. Foose, Franklin & Marshall College, Lancaster, Pa. (Theron W. Reed, Owens-Corning Fiberglas Corp., Newark, Ohio).

Mining Branch Council: H. M. Bannerman, U. S. Geological Survey, Washington, D. C. (E. P. Pfleider, University of Minnesota, Minneapolis, Minn.); E. H. Crabtree, U. S. Atomic Energy Commission, Grand Junction, Colo. (Ian Campbell, California Institute of Technology, Pasadena, Calif.).

Metals Branch Council: Robert F. Mehl, Carnegie Institute of Technology, Pittsburgh, Pa. (H. H. Kellogg, Columbia University, New York, N. Y.).

Petroleum Branch Council: Claude R. Hocott, Humble Oil & Refining Co., Houston, Texas (Paul R. Turnbull, Del Mar Drilling Co., Corpus Christi, Texas).

Council of Section Delegates, Executive Committee: Roger V. Pierce, Salt Lake City, Utah (H. R. Gault, Lehigh University, Bethlehem, Pa.

District 1, New York: James S. Vanick, International Nickel Co., New York, N. Y. (John H. Ffolliott, Miami Copper Co., New York, N.Y.).

District 2, Connecticut: W. E. Milligan, Hammond Metallurgical Laboratory, New Haven, Conn. (Leon W. Thelin, Chase Brass & Copper Co., Waterbury, Conn.).

District 3, St. Louis: Jack H. McWilliams, Aluminum Co. of America, Bauxite, Ark. (Carl H. Cotterill, American Zinc, Lead & Smelting Co., St. Louis, Mo.).

District 4, Pittsburgh: Hugo E. Johnson, Lake Superior Iron Ore Assn., Cleveland, Ohio (John C. Calhoun, Pennsylvania State University, State College, Pa.).

District 5, Chicago; R. A. Lindgren, International Harvester Co., Chicago, Ill. (C. T. Hayden, Sahara Coal Co., Chicago, Ill.).

District 6, Black Hills: James O. Harder, Homestake Mining Co., Lead, S. D. (Max W. Bowen, Golden Cycle Corp., Colorado Springs, Colo.).

District 7, Oregon: D. W. Johnson, Reynolds Metals Co., Troutdale, Ore. (Thomas J. Waters, Carbide & Alloys Co., Portland, Ore.).

District 8, Southern California: Basil Kantzer, Union Oil Co. of California, Los Angeles, Calif. (Ian Campbell, California Institute of Technology, Pasadena, Calif.).

District 9, Carlsbad Potash: George E. Atwood, Duval Sulphur & Potash Co., Carlsbad, N. M. (C. E. Presnell, International Minerals & Chemical Corp., Carlsbad, N. M.).

District 10, Delta: Byron L. Francis, Texas Co., New Orleans, La.

AIME Bessemer Committee to Meet

The Bessemer Steel Committee will meet on Wednesday, Jan. 12, 1955 at the Duquesne Club, Pittsburgh. The off-the-record meeting will be concerned with operating problems relative to Bessemer steelmaking and the problems of rolling Bessemer steel.

Announce 1955 Honors

The following honors will be conferred at the annual banquet on Feb. 16, 1955 in the Conrad Hilton Hotel in Chicago:

James Douglas Medal to Edwin Letts Oliver

Erskine Ramsay Medal to George Herman Deike, Sr.

Robert W. Hunt Medal to F. W. Boulger and R. H. Frazier

J. E. Johnson, Jr., Award to Gust Bitsianes

Rossiter W. Raymond Award to Harold Vagtborg, Jr.

Mathewson Gold Medal to W. G. Pfann

Robert H. Richards Award to Edward W. Davis

Peele Award to R. E. Thurmond and W. E. Heinrichs, Jr.

The Alfred Noble Prize Committee has announced C. S. Roberts, Dow Chemical Co., Midland, Mich., as the 1954 winner for his paper, *Creep Behavior of Magnesium-Cerium Alloys*, JOURNAL OF METALS, May 1954.

The following lecturers will also be guests of the Institute at the Head Table:

Institute of Metals Div. Lecturer, Clarence Zener; Howe Memorial Lecturer, John S. Marsh; and Jackling Lecturer, E. D. Gardner.

E. C. Gregg Addresses AIME Cleveland Section

At the October meeting of the AIME Cleveland Local Section, Prof. Earle C. Gregg of the physics dept., Case Institute of Technology, discussed the betatron and its application to industrial problems. Professor Gregg, who designed the Case betatron, briefly presented the fundamental principles of betatron design and operation. The many industrial, radiographic, and medical applications of this machine were discussed and illustrated. Following the meeting an opportunity was provided for examining the betatron and the instrumentation that is required for its operation.

Spokane To Be Scene Of 1955 Pacific Conference

The Pacific Northwest Regional Conference will be held in Spokane, Wash., Apr. 28, 29, and 30, 1955, according to Angus Y. Bethune, general chairman for the conference. The North Pacific, Oregon, and Columbia Local Sections will sponsor the meeting, with the Columbia Local Section acting as host. Institute divisions scheduled for participation are the minerals beneficiation, mining, geology, industrial minerals, extractive metallurgy, iron and steel, and physical metallurgy. A program for student participation is planned with one evening set aside for mineral industries education.

L. M. Pidgeon Receives Monell Medal

Lloyd M. Pidgeon, MBE of the University of Toronto, has been named recipient of the first Ambrose Monell Medal for distinguished achievement in mineral technology.

Mr. Pidgeon, head of the dept. of metallurgical engineering at the University of Toronto, is discoverer of the Pidgeon Process for production of metallic magnesium.

The first award committee consists of: Leo F. Reinartz, President, American Institute of Mining & Metallurgical Engineers; Charles C. Huston, President, Canadian Institute of Mining & Metallurgy; Brigadier R.S.G. Stokes, President, British Institution of Mining & Metallurgy; Felix Wormser, Assistant Secretary of the Interior; Professor Hassialis; Maxwell Gensamer; and Philip Bucky of the Columbia School of Mines.

Research in Progress To Be Reestablished

The Research in Progress session which was discontinued temporarily, will be reestablished at the Annual Meeting in Chicago in February 1955. Those who are interested in contributing to this meeting are invited to submit their contributions to E. O. Kirkendall, Secretary of IMD. Deadline for contributions is Jan. 1, 1955. Reports to be presented will be selected from those contributed.

Stainless Steel is a production steel

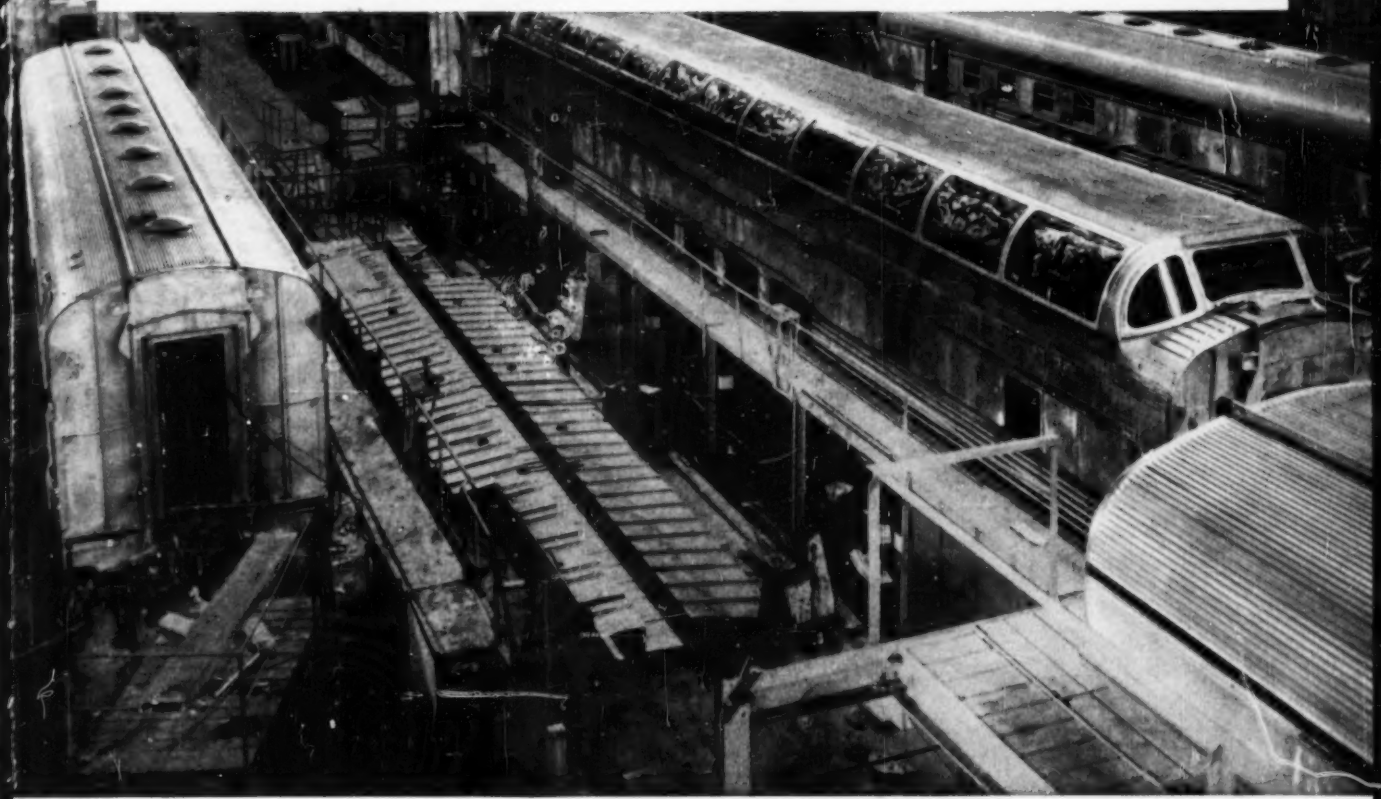


Photo Courtesy of The Budd Company

AT one time, the fabrication of Stainless Steel was looked upon as a specialty operation . . . one that could best be handled in limited quantities for custom jobs in small shops. Time and again, in plant after plant, this has been disproved.

Today, many plants making a variety of products and using a wide range of fabricating processes, are using Stainless as a *mass production* material—and using it successfully. As a result, a wide range of products from pots and pans to passenger cars are being made more attractive, more durable and more saleable with Stainless Steel on production lines.

One factor has usually played a vital role in such successful operations: an accurate knowledge of the characteristics and behavior of the particular Stainless used. For the word "Stainless" refers not to one steel but an entire family of special, high-quality steels. There are many grades with varying compositions and characteristics. Your fabricating operation and the end use of your product will dictate the grade to use most successfully and economically.

When you use USS Stainless Steel, you receive the greatest possible help along this line. Our representatives know the characteristics of the various grades; they'll help you pick the one that fits your operation and your product, show you how to use it most efficiently.

Keep this thought in mind: Stainless Steel isn't difficult to fabricate . . . it's just different. And it makes a better product.

UNITED STATES STEEL CORPORATION, PITTSBURGH • AMERICAN STEEL & WIRE DIVISION, CLEVELAND
COLUMBIA-GENEVA STEEL DIVISION, SAN FRANCISCO • NATIONAL TUBE DIVISION, PITTSBURGH
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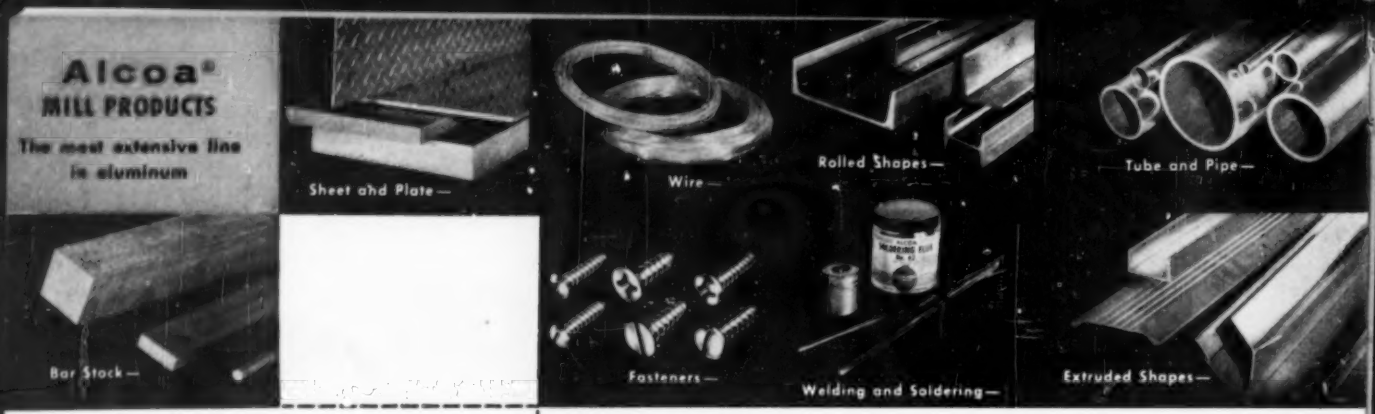
UNITED STATES STEEL EXPORT COMPANY, NEW YORK

USS STAINLESS STEEL

SHEETS · STRIP · PLATES · BARS · BILLETS — USS — PIPE · TUBES · WIRE · SPECIAL SECTIONS

4-1491

UNITED STATES STEEL



Alcoa manufactures a wide variety of standard rolled and extruded shapes for structural and architectural applications. With them, building costs are lower, construction time reduced.

There are extruded shapes for window sills, threshold, railings, copings and gravel stops; sheet for roofing and siding; pattern sheet for doors, partitions and siding; trim and moldings for glass-setting members, door accessories and interior moldings—and many more.

All these architectural products are available to you through Alcoa's 54 distributor outlets. They act as your warehouse, thus minimizing your stocking and handling problems; and they can readily supply your needs either in small lots for special jobs or in large quantities for continuous production runs.

To find out more about Alcoa's Architectural Products, call your local Alcoa Distributor. He is listed under "Aluminum" in the classified section of your telephone directory. Or write ALUMINUM COMPANY OF AMERICA 878-M Alcoa Building, Pittsburgh 19, Pennsylvania.

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ALCOA ALUMINUM

ALUMINUM COMPANY OF AMERICA

He's using the telephone that lends an extra hand

"Sure . . . I'm looking up the figures right now!" From busy executives to clerks, people in business can work more easily and efficiently with Bell's new Distant Talking Telephone. Small white rectangle is the loudspeaker.

For people who want to keep *both* hands free when they telephone, Bell Telephone Laboratories engineers have devised a new telephone with a sensitive microphone in its base.

To use it, simply press a button. The microphone picks up your voice and sends it on its way. Your party's voice comes to you through a small loudspeaker. Both hands are left free.

The volume can be adjusted to suit yourself. If privacy is needed, you simply lift the handset; this shuts off the microphone and loudspeaker and you talk just as you would on a regular telephone.

This new development of Bell Laboratories increases the number of ways your local Bell telephone company can serve in businesses and homes.

Pencil points to microphone in base of new telephone. Left-hand button controls volume, center one turns set "on" and lights up while in use. The third is an "off" button.

Bell Telephone Laboratories

Improving telephone service for America provides careers for creative men in scientific and technical fields.

Personals

F. C. T. DANIELS

Fred C. T. Daniels has retired as vice president of research and development for Mackintosh-Hemphill Co., Pittsburgh. Mr. Daniels joined Mackintosh shortly after his graduation from college. He has served as a vice president for the past 26 years. Mr. Daniels will serve in a consulting and advisory capacity.

James F. McQuillan, plant manager, American Smelting & Refining Co., federated metals div., has been transferred from Whiting, Ind., to Houston, Texas.

R. B. Wittenberg is vice president and general manager, electrode div., for Great Lakes Carbon Corp., New York, N. Y.

Sumio Yukawa has joined the General Electric Co., materials and processes laboratory, Schenectady, N. Y.

R. N. Millar has resigned from the British General Electric Co., Sydney, Australia, and accepted a position with Fraser & Chalmers Engineering Works, Erith, Kent, England.

William B. Wallis, president Pittsburgh Lectromelt Furnace Corp., Pittsburgh, has been elected president of the Foundry Equipment Mfrs. Assn. **David E. Davidson,** vice president, sales, Link-Belt Co., Chicago, was elected vice president.

James C. Gray was appointed vice president—operations—coal, U. S. Steel Corp., Pittsburgh. Mr. Gray had been manager of manufacturing operations, Tennessee Coal & Iron Div. In his new position he will head the company's coal mining activities in Pennsylvania, West Virginia, and Kentucky.

John A. Michelson has joined the Hinchman Corp., Detroit, as corrosion engineer.

W. H. SPARR, JR.

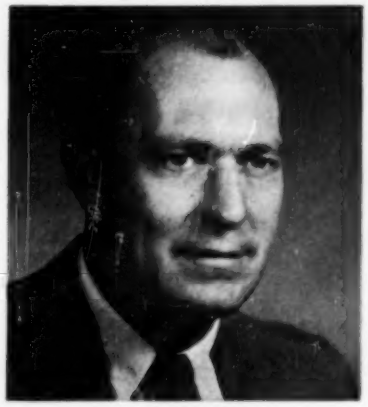
William H. Sparr, Jr., head of development and research div., technical field section, International Nickel Co., Inc., Pittsburgh, has been transferred to New York and named in charge of the steel section of the nickel sales dept. Mr. Sparr has been associated with Inco since 1946.

Kent R. Van Horn, director of research, Aluminum Co. of America, presented the Campbell Memorial Lecture of the ASM at the Metal Show in Chicago.

Seymour J. Sindeband has been elected president of the Teleregister Corp., Stamford, Conn. Mr. Sindeband had been president of Mercast Corp., and its affiliate Alloy Precision Castings Co.

James H. Moore has been appointed general manager of Vacuum Metals Corp., Syracuse, N. Y. Mr. Moore joined National Research Corp. in 1946 as research and development metallurgist; in 1948 he became director of the metallurgical dept.

J. H. MOORE



R. E WARRINER

Reuel E. Warriner, in charge of the steel section of the International Nickel Co. sales dept., has been appointed vice president, sales, Climax Molybdenum Co., New York. Mr. Warriner had been associated with Inco for more than 19 years. **Robert S. Archer** has been appointed chief metallurgist for Climax.

Ward W. Minkler, formerly Pacific coast manager of sales and technical service, has been named assistant manager of market development of the Titanium Metals Corp. of America, New York. Prior to joining Titanium Metals, Mr. Minkler was a member of the metallurgy staff for Babcock & Wilcox Co., tubular products div., Beaver Falls, Pa.

Henry J. Fisher, formerly a scientist on the staff of the Dept. of Mines & Technical Surveys, Ottawa, Canada, has joined the General Electric Research Laboratory, Schenectady, as a research associate.

P. W. Bakarian is with Cramet, Inc., Chattanooga, Tenn. He had been associated with the Crane Co., Chicago.

William K. Whiteford, president, Gulf Oil Corp., Pittsburgh, has been elected to the board of directors of Jones & Laughlin Steel Corp., Pittsburgh, Pa.

Paul Zeigler, director of Kaiser Aluminum & Chemical Corp.'s dept. of metallurgical research, Spokane, Wash., will be located at company headquarters, Oakland, Calif. **D. W. Smith,** associate director of the dept. of metallurgical research, Spokane, will assume the duties of laboratory manager in addition to those of his present post. **S. E. Maddigan,** head of the mechanical evaluation section, has been made assistant director of the dept.

fluidity and rare earths

Many steel men have long recognized that if they could pour steel on "its cold side," they would get better rolling qualities. Unfortunately, steel being as it is, this procedure often results in dirty steel or ladle skulls.

MCA's Rare Earths, commercially available as RareMeT Compound, increase the fluidity of liquid steel and therefore aid in allowing lower tapping temperature. At the same time, RareMeT additions make possible clean steel at lower temperatures since impurities can free themselves from this high fluidity steel.

RareMeT Compound is most effective at lower temperatures. Therefore both better rolling and cleaner steel can be expected with this addition. Lower costs are easily achieved when RareMeT Compound is properly used—better rollings—higher yields—lower surface preparation costs.

Requests for additional technical data for specific problems are welcomed. Complete and immediate response is given to all requests.

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Obituaries

Appreciation of
J. Hunter Nead
By Leo F. Reinartz

J. Hunter Nead, at the age of 67, passed away suddenly at Dunes Acres, Chesterton, Ind. on Sept. 13, 1954.

Mr. Nead was born in Kansas City, Mo. on Mar. 10, 1887.

He graduated from the University of Michigan with a Bachelor of Science degree in chemical engineering.

His was a long, honorable industrial career. He started as chief metallurgist at Franklin Motor Car Co., then successively he became engineer of tests at the Watertown Arsenal and chief metallurgist for the Minneapolis Steel & Machinery Co.

In 1919 he came to the American Rolling Mill Co. (now Armco Steel Corp.) as metallurgist and served acceptably with that company until 1931. During those years he assisted in making experiments, which at a later date led to the development of new steels for the manufacture of automobiles.

From 1931 until he retired in April 1952 he was associated with the Inland Steel Co., East Chicago, Ind. as chief metallurgist and later as manager of the metallurgical and inspection depts. All of his associates have spoken highly of his work and his relations with other supervisors and associates. In both of his most recent positions he worked with distinction and usefulness to his company and the steel industry.

He served his country as Captain in the Ordnance Dept., U. S. Army in 1918 and 1919.

He was a member of many technical societies—American Society for Metals (past director), American Iron and Steel Institute, British Iron and Steel Institute, British Institute of Metals, Assn. of Iron and Steel Engineers, American Society for Testing Materials and others. For many years he was a member of the University Club of Chicago.

Mr. Nead was a valued member of the American Institute of Mining & Metallurgical Engineers. For several years he was active in the National Open Hearth Conference of the Institute. In 1939 he served as Chairman of the Iron and Steel Div.

In 1938 he and his colleague, T. S. Watson, received the Robert W. Hunt Award given by the Institute for an outstanding and original contribution to iron and steel literature.

Hunter, as he was affectionately known by his many friends at Armco, Inland and in the steel industry in general, was married to Martha Ohnstrand on Dec. 9, 1913. They have one daughter, Marcia, married to G. B. Lanman, Jr. Hunter had three grandchildren who were near and dear to him.

Since his retirement, his wife and he lived quietly in Florida during the winter months and together they enjoyed their home on the dunes during the summer season. He was in the best of spirits prior to his demise. In recent years he became an avid collector of sea shells and had a large, beautiful collection. Prior to his illness he was reputed to have been an expert ping pong player and I can vouch that he was a golf enthusiast.

Hunter will long be remembered by his many acquaintances as a sincere, loyal, warm hearted friend and co-worker.

As President of AIME I salute the contributions of an outstanding metallurgist and gentleman.

Appreciation of
George H. Rupp
By Charles E. Golson

George H. Rupp (Member 1927) died Oct 11, 1954 in Pueblo, Colo., after a lingering illness. He was manager of mines of the Colorado Fuel & Iron Corp.

George was born on Nov. 10, 1889 in Bessemer, Mich., and even before college days worked in the Colby, Ironton, and Yale iron mines in Bessemer. Graduating as valedictorian from high school, he first attended the University of Michigan with the idea of becoming a doctor. After a year of pre-med, upon his father's death, he transferred to the Michigan College of Mines. He graduated at the top of a class, in 1913, which has produced quite a number of prominent men, and with his degrees of B.S. and E.M. became engineer for the Gogebic County Road Commission. For two years he was efficiency engineer for the Newport Mining Co. at Ironwood, Mich. Later, he and L. C. Brewer, as partners, operated an independent ore exploration company. This company reopened the old Norrie mine with Rupp as superintendent. This was still in operation as late as 1936 under the new name of the Townsite mine. At this time Rupp also investigated a new discovery of iron ore in the Belcher Islands of Hudson Bay and spent a year in the Far North country living among the Indians and Eskimos.

In 1922, Rupp was employed by the Ford Motor Co. and became superintendent of the Imperial mine at Michigamme, Mich. In 1927, he became associated with Calumet & Hecla Mining Corp. at Calumet, Mich., in charge of mining operations, which employment lasted until 1929.

In 1929, George Rupp joined Colorado Fuel & Iron Corp. as general superintendent of iron mines and quarries. In 1930, the mining department, to include all raw materials produced by C.F.&I., was formed and to his death, George was manager of this department.

George also held two patents, together with an associate, on coal washing and dewatering machines.

George had married Miss Zita Johnson, whom he had beaten out of the honor of valedictorian at high school commencement by 0.2 pct in grades. Besides his widow, he is survived by two daughters and two sons.

George, with his varied career and wide experience, was an outstanding engineer and administrator. He was extremely friendly to youth, and cooperative to all those who met him professionally or in private life; his wit and humor livened the stories of his experience. He was extremely interested in AIME and did everything in his power to interest young engineers in joining the Institute. He leaves a wide circle of friends and associates who will deeply regret his death.

Walter F. Rittman (Member 1919) died Sept. 26, 1954. He was head of the engineering dept. of Carnegie Institute of Technology from 1921 to 1933 and a pioneer in the field of chemical engineering of petroleum.

Mr. Rittman, in recent years a consulting engineer in Pittsburgh, was born in Sandusky, Ohio, in 1883. A graduate of Swarthmore College, he received his Ph.D. from Columbia in 1914. He was with the U. S. Bureau of Mines from 1914 to 1921.

Later Mr. Rittman served as consulting engineer for the State of Pennsylvania and the U. S. Dept. of Agriculture. He was the author of numerous articles dealing with application of physical chemistry for industrial processes, especially those dealing with fuel, oil, and gas. He was past national president of the Society of Industrial Engineers, a fellow of the American Academy for the Advancement of Science, and a trustee of Ohio University.

Stanley Barber (Member 1941) was first employed in industry by the firm of Herman Waller, Vienna. He later moved to England and dealt in nonferrous minerals and metals. In 1920 Mr. Barber joined the staff of P. Blasbal Co., Cologne as a buyer for minerals and ores in England and abroad. In 1936 he obtained a position with N. V. Midden Nederlandsche Handelsmaatsschappiy, Amsterdam as their representative in London. When Holland was invaded during World War II, Mr. Barber became a representative for Overseas Metal & Ore Corp., New York, in England and overseas. At the time of his death he was residing in New York City.

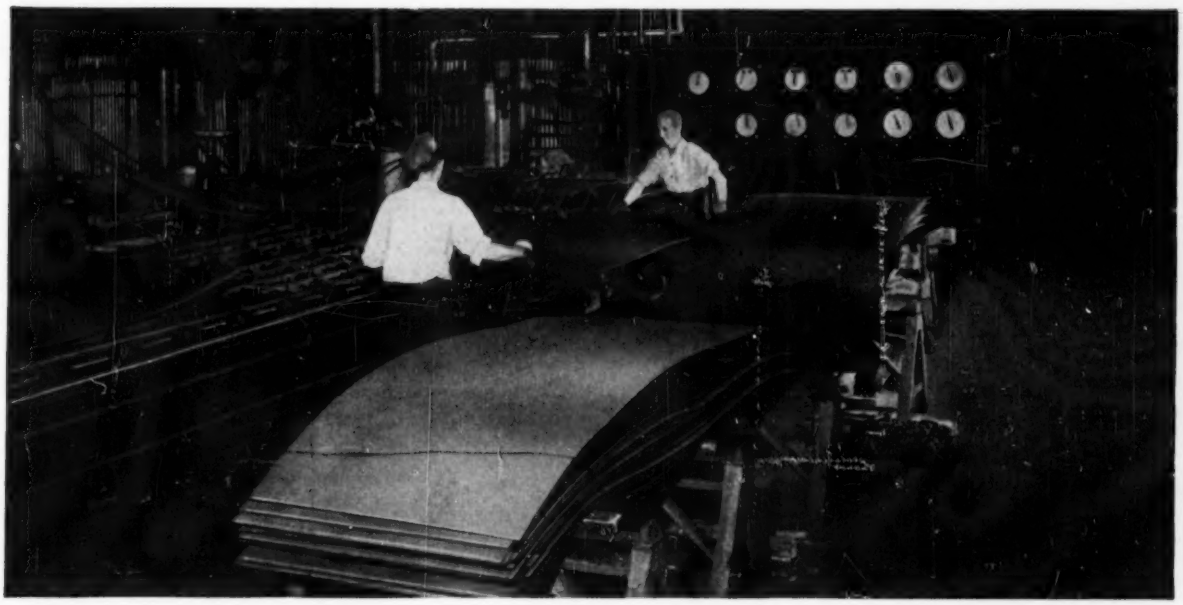
Charles M. Paul (Member 1954), died on Sept. 1, 1954. Mr. Paul had been associated with the Bethlehem Steel Co., Johnstown, Pa., as general foreman.

Charles E. Sumpter (Member 1951) died on Sept. 13, 1954. Mr. Sumpter

(Continued on p. 1426)

Flexible control

assures precision in annealing stainless sheet at Atlas Steels



ATLAS STEELS, LTD., one of the British Commonwealth's largest producers of quality steels, makes good use of modern Brown instrumentation in the production of many stainless, high speed and special purpose alloys. The accurate, flexible regulation of temperatures and fuel-air ratio they obtain by custom-engineered Brown control systems plays an important part in maintaining consistently high quality of product and economical utilization of fuel.

The big sheet annealing furnace pictured above is equipped with coordinated control for each of four zones. For each, an *ElectroniK* potentiometer actuates the combustion air valve to hold constant temperatures. An air flow controller meters combustion air flow and resets the control index of a Brown flow ratio controller. This third instrument, in turn, regulates the gas or oil fuel valve . . . holding fuel flow in accurate proportion to air at all firing rates. The net result . . . highly accurate temperature control and top fuel economy.

Whether you're installing new furnaces or modernizing old ones, it will pay you to investigate Brown instrumentation. Your nearby Honeywell sales engineer will be glad to discuss your specific problem . . . and he's as near as your phone.

MINNEAPOLIS-HONEYWELL REGULATOR CO., *Industrial Division*, Wayne and Windrim Avenues, Philadelphia 44, Pa.

● **REFERENCE DATA:** Write for Catalog 1531, "*ElectroniK* Controllers", and for Data Sheet No. 4.3-4a, "Fuel-Air Ratio Controllers."

MINNEAPOLIS **Honeywell**

BROWN INSTRUMENTS

First in Controls

was a graduate of the University of Pittsburgh. Following graduation, he joined the Homestead Works, Carnegie Illinois Steel Corp., as a metallurgical observer. In 1943 he was promoted to chief observer for the Farrell Works. Mr. Sumpter joined U. S. Steel Corp. in 1945 as an associate physical chemist. In 1947 he became a ceramic engineer for the South Works of Carnegie. He was made general supervisor of ceramic development in 1950. At the time of his death Mr. Sumpter was residing at Lamsing, Ill.

Nicholas Alexander Ziegler (Member 1923), active for many years in metallurgical research and development, died on Aug. 20, 1954. Due to failing health Mr. Ziegler, during the past year, had relinquished many of his responsibilities as supervisor of metallurgical research for the Crane Co., Chicago, with whom he had been associated since 1936. Mr. Ziegler was born in Russia, came to this country during World War I, entered Columbia University, and graduated with an A.B. in 1921. He continued in the Engineering School and earned a degree in metallurgical engineering in 1924. After leaving the University, he worked for a year as a junior metallurgist for the Crucible Steel Co. of America. In 1925 he joined the metallurgical research staff of the Westinghouse Electric & Mfg. Co., Pittsburgh, where he remained until 1934. From 1934 to 1936 much of his time was spent on metallurgical consulting work in the Pittsburgh and Cleveland districts. In parallel with his other activities, Mr. Ziegler was part time instructor in metallography in the evening school of Carnegie Institute of Technology from 1925 to 1936. He also was instructor for a course at the Illinois Institute of Technology for the 1948 to 1949 term.

Mr. Ziegler was well known for his research in vacuum fusion analysis of gases in metals, magnetic properties of iron, mechanical properties and structural characteristics of various alloy steels and cast irons. He received the H. M. Howe Medal in 1953 from the American Society for Metals. He was also granted about eight patents. His latest achievement was pioneering a titanium development program at Crane Co.

During World War II, Mr. Ziegler was associated part time with the Office of Scientific Research and Development. As a result of this work, he was awarded a citation in June 1947. He also received a Naval Ordnance Award in June 1946. He was a fellow in the American Assn. for the Advancement of Science. He was registered among the American Men of Science, and was a member of the AIME, ASM, and AFS. Amateur painting was Mr. Ziegler's best liked hobby. In 1950 he won an award in the National Amateurs Painting contest.

Necrology

Date Elected	Name	Date of Death
1936	Charles J. Abrams	Oct. 17, 1954
1954	Roy M. Atwater	Oct. 6, 1954
1948	Isaac G. Brown	Aug. 24, 1954
1903	H. H. Claudet	Aug. 8, 1954
1947	Jackson Coffin	Sept. 21, 1954
1936	Arthur L. Harris	Oct. 2, 1954
1941	John Mills Hoff	July 22, 1954
1918	Robert Job	June 1951
1944	Frederick John Kasper	Oct. 19, 1954
1935	F. S. McNicholas	Sept. 1954
1938	William S. Pugh	Jan. 19, 1954
1927	George Herbert Rupp	Oct. 11, 1954
1935	S. Joseph Swainson	Oct. 22, 1954
1950	William T. Turrall	Sept. 22, 1954
1919	Walter F. Rittman	Sept. 26, 1954
1948	M. B. Willey	Unknown

Proposed for Membership
— Metals Branch AIME —

Total AIME membership on Oct. 31, 1954 was 21,569; in addition 1782 Student Associates were enrolled.

ADMISSIONS COMMITTEE

O. B. J. Fraser, Chairman; R. B. Caples, Vice-Chairman; F. A. Ayer, A. C. Brinker, R. H. Dickson, Max Gensamer, Ivan A. Given, Fred W. Hanson, T. D. Jones, Sidney Rolle, J. H. Scaff, John T. Sherman, F. T. Sisco, Frank T. Weems, R. L. Ziegfeld.

The Institute desires to extend its privileges to every person to whom it can be of service, but does not desire as members persons who are unqualified. Institute members are urged to review this list as soon as possible and immediately to inform the Secretary's office if names of people are found who are known to be unqualified for AIME membership.

In the following list C/S means change of status; R, reinstatement; M, Member; J, Junior Member; A, Associate Member; S, Student Associate.

California
Oakland—Rose, Harry B. (M)

Connecticut
Riverside—Lee, Lewis V. (M)

Illinois
Chicago—Booth, Philip H. (M)
Chicago—Snyder, Milton H. (M)
Chicago—Whittenberger, Edmond J. (M) (C/S—A-M)

Indiana
Hammond—Ferguson, Scott V. (M) (R. C/S—J-M)
Highland—McGaughey, Emil R. (M)
Michigan City—O'Hara, Bernard N. (A)
Whiting—Flowers, Richard R. (J)

Maryland
Halethorpe 27—Ross, Roy R. (J)

Massachusetts
Newton Centre—Adams, Clyde M., Jr. (J)

Michigan
Detroit—Solomon, Alexander (J)

Minnesota
Minneapolis—Prasky, Charles (M)

New York
Niagara Falls—Buttner, Frederick H. (M) (R. C/S—S-M)
Niagara Falls—Fountain, Richard W. (J)
Troy—Jemian, Wartan A. (J)

Ohio
Cincinnati—Robertshaw, Fred C., Jr. (M)
Middletown—Merritt, John C. (J)

Oklahoma
Tulsa—Westphal, Frederick A. (M)

Pennsylvania
West Lawn—Culp, Neil J. (J)

Tennessee
Copperhill—Harbison, Joe G. (M)

Virginia
Alexandria—Weertman, Johannes (M)
Richmond—Matthieu, Donald E. (M)

Washington
Richland—Sanderson, Morris J. (M)

Wisconsin
Milwaukee—Matters, Robert G. (M)
South Milwaukee—Holm, Paul J. (A)

Java
Djakarta—Surapradja, Raden A. (M)

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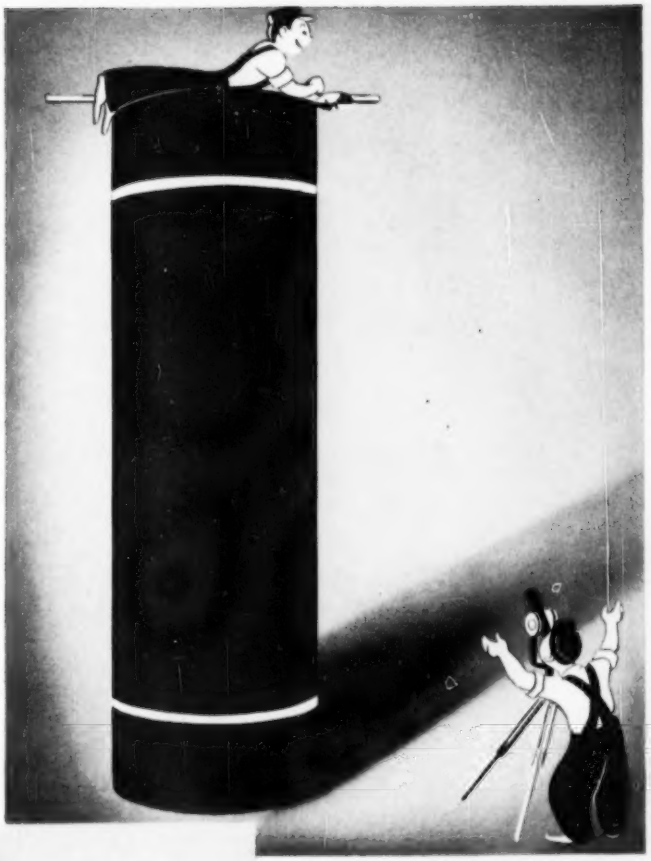

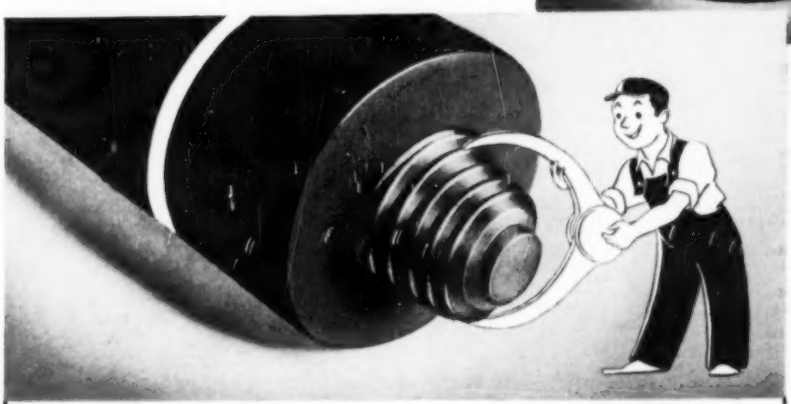

LOOK AT NIPPLE SIZE. Electrode nipples are another potential source of economy and improved performance. For example, if you use electrodes of 16" diameter or larger, and you are not already using the smaller nipples *pioneered by* NATIONAL CARBON COMPANY, you may be able to make this switch and save money while getting even stronger joints due to the thicker socket-walls provided by the smaller nipple sizes.

LOOK AT ELECTRODE LENGTH. Frequently we are able to direct customers to valuable economies in cost-per-pound of electrode, as well as to other cost-saving advantages, simply by recommending a longer length than they are already using. For instance, when you switch from a 60" electrode to one of 72", you reduce the number of electrodes, handled and used, by *one in six*, and, even more important, you cut down the number of joints made by the same high percentage. Obviously, this increases furnace-availability and boosts production.

• These are only two of many ways that NATIONAL CARBON'S electrode technical-service facilities have helped users get the most for their electrode dollar. Let your NATIONAL CARBON representative survey your electrode and nipple requirements. He may help you get substantial savings and improved electrode performance.

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NATIONAL CARBON COMPANY

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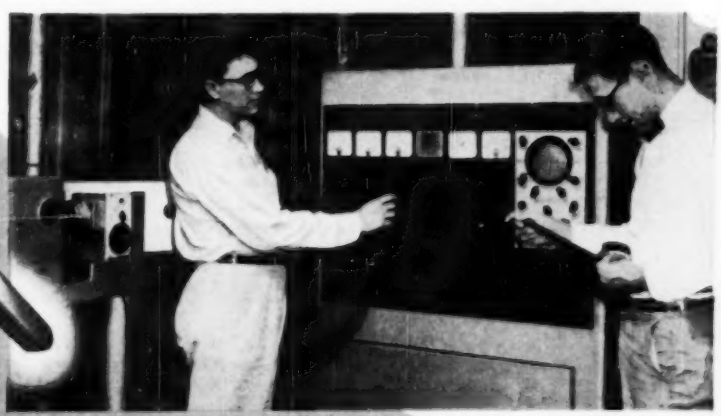
77-NI...

PROVEN HIGH CHROMIUM ALLOY GRATE BARS
PRODUCED BY **CARONDELET'S** MELTING *and* CASTING METHODS
...*and* **NOW** FURTHER CONTROLLED BY
SPECTROGRAPHIC ANALYSIS . . . . . .

...MEANS GRATE BARS FOR FERROUS *and* NON-FERROUS SINTERING WITH AN INCREASED SERVICE LIFE OF FROM 6 TO 10 TIMES . . . . . .

77-NI alloy Grate Bars are receiving more and more recognition as a means of cutting sintering costs. CARONDELET'S engineering, metal melting procedures and now spectrographic analysis of "pre-pouring" Test Bars, as shown, are reasons for the superiority of 77-NI.

Heat, wear and shock resistance are dependent upon close metal analysis control. The Spectrograph gives us the opportunity to analyze Test Bars from each heat quickly and adjust before pouring, thus assuring the best chemical properties in the final sintering grate bar castings. 77-NI grate bars cost less because they last longer.



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